MINING ENGINEERING

NOVEMBER, 1951

Circular Steel Sets Prove Economical

Nitrate

This new Model "K" WILFLEY pump, part of an installation in a Chilean nitrate plant, is making a splendid record of efficient, cost-saving operation in handling nitrate slurry.

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centrifugal PUMPS

Companion to the famous
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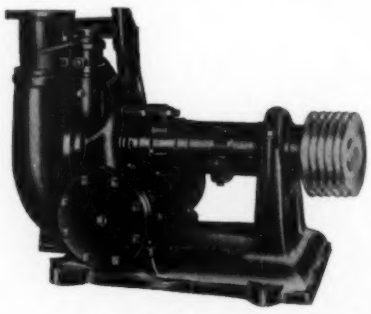

Worth-while power-savings and stepped-up 24-hour-a-day production result from important mechanical improvements and refinements embodied in the new Model "K" WILFLEY Sand Pumps. These pumps are noted for dependability, economy and trouble-free performance. Easy interchangeability of wear parts. Low maintenance. Continuous, 24-hour operation without attention. Individual engineering on every application. Write or wire for Model "K" Bulletin 200.

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MINING ENGINEERING

Incorporating Mining and Metallurgy, Mining Technology and Coal Technology

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FEATURES

ARTICLES

TRANSACTIONS

COVER: The ground condition shown in the photograph makes it necessary to make frequent changes of timber sets. The circular steel set, diagrammatically shown is the method of holding such ground chosen by Miami Copper Co. The results are ably described in the article on p. 940 by J. W. Still, general superintendent at Miami. The photograph is courtesy of Bethlehem Steel Co.

Personnel Service

THE following employment items are made available to AIME members on a non-profit basis by the Engineering Societies Personnel Service, Inc., operating in cooperation with the Four Founder Societies. Local offices of the Personnel Service are at 8 W. 40th St., **New York 18;** 100 Farnsworth Ave., **Detroit;** 57 Post St., **San Francisco;** 84 E. Randolph St., **Chicago 1.** Applicants should address all mail to the proper key numbers in care of the New York Office and include 6c in stamps for forwarding and returning application. The applicant agrees, if placed in a position by means of the Service, to pay the placement fee listed by the Service. AIME members may secure a weekly bulletin of positions available for $3.50 a quarter, $12 a year.

POSITIONS OPEN

Geologist, young, with from two to three years' experience, to do some field work as well as industrial contact. Considerable traveling. Salary, $4800 a year. Headquarters, North Carolina. Y6092.

Product Development Engineer, 28-40, engineering degree, some experience in industrial marketing, for post emergency product development. Company is equipped to do a wide variety of metal fabrication work including press work, metal finishing and assembly; has a large machine shop, engineering design and drafting organization. Salary open. Location, Ohio. Y6087-CD.

Mine Foreman experienced in mining especially square setting, cut and fill and top slicing. Will assist mine superintendent and take over the mine if necessary. Should have considerable experience in timbering. Will spend about half day on mine work and the rest of the time on engineering. Salary, $4200 to $4500 a year, plus room and board if single man; if married, will receive a furnished house and $75 a month allowance. Location, Honduras. Y6130.

Expediter, 35-66, experienced in expediting purchases of materials and equipment for mining and/or metallurgical operations; purchasing materials and equipment for such operations and expansion of plant facilities therefore; technical sales of machinery and equipment used in non-ferrous metals industry; technical sales of any machinery and equipment used in mining or heavy industry. Must be willing to travel around the country. Must understand paper work of buying organization, and also of manufacturing and fabricating organizations. Knowledge of government regulations, i.e., priority and allocation controls helpful. Salary open. Location, New York, N. Y. Y6011.

Concentrator Metallurgist, 30-35, graduate in ore dressing with practical experience, to take charge of laboratory testing for large mining company. Spanish helpful. Three-year contract, transportation, living quarters. Location, South America. Y6010.

Civil or Mining Engineer, 30-40, graduate, with some experience in road building and construction for field work. Knowledge of Spanish essential. Location, Caribbean area. Y6000.

WANTED: Large company seeking recent graduate mining engineers interested work Mexico. Single men provided furnished room rent free, married men adequate rent free furnished housekeeping accommodations. Work involves surveying surface underground, mapping, sampling, geology. Opportunity subsequent transfer operating work in case of preference. Send full particulars to:

Box J-22 MINING ENGINEERING

Mine Master Mechanic, 40-45, preferably married, with foreign experience, to supervise mining and power equipment maintenance covering diesel power plant, compressors, loaders, drills, steels, etc. Air transportation; two-year contract. Knowledge of French desirable. Salary, $6000 a year plus quarters. (b) Junior Mining Engineer for underground duties. Salary open. Location, Africa. Y6019.

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Metallurgical Engineer, 25 years' experience ore concentration, process development and extractive metallurgy while associated with prominent American and South African companies. An authority on certain phases of ore treatment. Desires position as chief metallurgist or consultant for mine or metallurgical interests preferably in western U.S.A. M-648.

Graduate Mining Engineer and Executive, with wide experience in charge of mining and ore dressing operations, seeks change of position within the United States, with reputable company. Record includes efficient, low cost operations and favorable labor relations. M-649.

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Letters to The Editor

Cylindrical Not Conical!

I have been a member of the AIME for many years and have always considered information written in its publications as authentic and based on actual facts. I am therefore surprised at the diagram designation of some of the equipment in the Nickel recovery flow sheet in the August MINING ENGINEERING issue, page 671.

There is only one mill made of the shape indicated by the diagram of the ball and rod mills in the flow sheet. If the sketch had shown a cylindrical ball and rod mill it could have been any one of a number of manufacturers' mills and, in that respect at least, would have been correct.

The diagram designation is fallacious and exclusive to one manufacturer. The Mine and Smelter Supply Co. sold the first Marcy mills to International Nickel Co. in 1925 and since that time, to the best of my knowledge, they are using no other mill than Marcy ball mills and Marcy open end rod mills in their concentrators. At Copper Cliff, the new Creighton mill and at Huntington, West Va., they are using 57 Marcy ball and rod mills. These mills represent many repeat orders over a period of 25 years.

Many engineers not familiar with the International Nickel concentrator operations may believe the diagrams on the nickel recovery flow sheet are correct and be misinformed as to the type of equipment being used.

I would, therefore, appreciate correction as to the type of grinding equipment used.

C. G. WILLARD, MANAGER
MARCY MILL DIV.
MINE & SMELTER SUPPLY CO.

Hope this gets the record straight.—Editor

Can You Top This?

Probable but not true. [This terse comment captioned the letter of Cleveland-Cliffs' F. J. Haller, August MINING ENGINEERING, p. 642, in which he challenged our statement in the article "Opening the Pyne Mine of the Woodward Iron Co.", December MINING ENGINEERING, p. 1230, in which we claimed that the Pyne mine, hoisting about a million tons a year from one shaft, was "probably" the world's biggest producer. Mr. Haller mentioned that the Mather A shaft produced 1,251,963 tons.]

Being only a little timid Swede, at present living in this country, it sometimes is a little difficult to hide a smile about you Yankees always desiring to be the biggest in the world.

Here a Cleveland-Cliffs man is telling a Pyne mine man that his operation is bigger than Pyne mine and as a consequence, of course, the biggest in the world.

By the way, do you Americans ever look over the world when discussing the word "biggest?" If not, let me inform you that Graugesberg in Sweden is hoisting 1,900,000 tons a year in a single shaft and will hoist 2,300,000 tons next year.

I don't know whether that is the biggest in the world, besides we Swedes don't care—but I can't help it, we still like the Yankees.

A. KJELGAARD
MILWAUKEE, WIS.

We like you too.—Editor

More on Traverse Tables

Whilst *Gurden's Traverse Tables* is not in vogue in this country, other traverse tables have nevertheless been used, but one still has to rely on the good old-fashioned method of computation using calculators and natural trigonometric functions, for very accurate and precision surveys. [Reference is made to "Computing Survey Notes with Traverse Tables" by J. Paul Trueblood, July MINING ENGINEERING, p. 583.]

The comparisons below illustrate the difference in the values for angles measured:

Example 1—Measured distance of 1494 ft with a bearing of 89° 49'.

	Lat.	Dep.
a) Using Trig. functions:	4·78044987654	1493·99276
b) Using Gurden's tables:	4·7808	1493·9960
	+0·00035012346	−0·00326

Example 2—Measured distance of 1494 ft with a bearing of 89° 48' 30".

	Lat.	Dep.
a) Using Trig. functions:	4·99774	1493·99207
b) Using Gurden's tables and obtaining values of seconds by interpolation:	4·99945	1493·99469
	0·00171	−0·00262

. . . the difference in the corresponding values would apparently be very unsuitable for comprehensive precision surveys.

Just for the record, if a '1 second Watts Theodolite' was used, the values for angles measured and computed by interpolation, would not be too accurate.

RUSI D. MOGRELIA
NATIONAL COAL BOARD
YORKS, ENGLAND

Who's On Top?

In September 1951 issue of MINING ENGINEERING is an article on uranium ["Uranium—Swords or Plowshares," p. 762].

Out of it I do not gather who the three leading producers of uranium are in the world; neither am I certain of whether uranium or uranium oxide (U_3O_8) is the proper unit by which to evaluate uranium ore.

Along this line the article mentions Africa as being "World's largest producer of high grade uranium ore." However, the largest producer of ore which can mean tons, does not convey to me that by that Africa is the world's largest producer of uranium oxide (or uranium).

Canada is also quoted as being the third largest producer of uranium ore. By that is it meant of uranium oxide? I assume that the United States is the second largest producer.

R. L. HEALY, GENERAL MANAGER
WRIGHT-HARGREAVES MINES, LTD.
KIRKLAND LAKE, ONT.

U_3O_8 is the proper unit of measure of uranium content of ores and the Belgian Congo, United States, and Canada rank first, second, and third, respectively. When the recent discoveries at Beaver Lodge Lake come in, Canada will probably step up into the number two spot.—Editor.

Unrecorded Mineral Curricula

I note with interest the directory of mining engineering schools in United States and Canada published in the August 1951 issue of MINING ENGINEERING. May I point out that the Wisconsin State Legislature has authorized the Wisconsin Institute of Technology to give 4-year courses and B.S. degrees in mining engineering and civil engineering, beginning September 1951. Under the old 3-year course system, Wisconsin Institute of Technology turned out in the neighborhood of 30 to 40 graduates a year, most of whom completed schooling for their degrees at some other institution. Thus we anticipate granting a substantial number of degrees from this institution in the future.

W. A. BROUGHTON, REGISTRAR
THE STATE OF WISCONSIN
INSTITUTE OF TECHNOLOGY

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MEET THE AUTHORS

D. H. GIESKIENG

D. F. KAUFMAN

H. RUSH SPEDDEN

K. P. WANG

A. M. Gaudin *(Progeny in Comminution, P. 969)* was born in Smyrna, Turkey of French parents. He attended Columbia University and Montana School of Mines. He received an E.M. degree from Columbia in 1921 and an honorary Sc.D. degree from Montana School of Mines in 1941. He was professor of mineral dressing at University of Utah, research professor of mineral dressing at Montana School of Mines, and Richards professor of mineral engineering M.I.T. He is at present director of M.I.T. research laboratory. Besides being a member of AIME, he also holds membership in American Chemical Society, Canadian Institute of Mining Engineers and Society of Civil Engineers of France. Mr. Gaudin is well known to our readers as he has presented about 45 papers before AIME. Fishing, painting, photography and collecting stamps are his hobbies.

D. H. Gieskieng *(Jaw Crusher Capacities, Blake and Single-Toggle or Overhead Eccentric Types, P. 971)* was born in Denver, Colo. and attended the Colorado School of Mines. He received a metallurgical engineering degree. Mr. Gieskieng has worked for Golden Cycle Corp. in Colorado Springs, Colo. and for Allis-Chalmers Mfg. Co. Instrumentation is of particular interest to Mr. Gieskieng. Besides being a member of AIME, Mr. Gieskieng is also a member of the Colorado Metallurgical Society. Mr. Geiskieng has presented a previous paper before the AIME in San Francisco in 1949. Amateur radio and photography are his favorite hobbies.

D. F. Kaufman *(co-author with Messrs. Gaudin and Spedden)* attended Massachusetts Institute of Technology and received his B.S. in June 1951. Mr. Kaufman was research assistant at mining engineering laboratory at M.I.T. in summer of 1950 and research assistant in ceramics laboratory in M.I.T. in summer of 1951. At present time he is teaching assistant in process metallurgy laboratory at M.I.T. Electrolytic production of titanium is of particular interest to Mr. Kaufman. Sailing, mountain climbing and photography are his favorite hobbies.

J. W. Still *(Circular Steel Sets, P. 940)* was born in Tucson, Arizona and attended the University of Arizona. From 1921-1930 he worked for Miami Copper Co. as a transitman and later as stope and development engineer. He was employed as mine superintendent and general manager from 1936 to 1944 for Bagdad Copper Corp. He spent 1944 and part of 1945 in Washington, D. C. with MRC and FEA on wartime government metal purchasing. Since June 1945 he has been with Miami Copper Co. as mine superintendent and at present is general superintendent. Problems of special interest to Mr. Still is underground mine mechanization. An AIME member, Mr. Still's favorite hobby is hunting.

H. Rush Spedden *(co-author with Messrs. Gaudin and Kaufman)* attended Lewis and Clark High School in Spokane, Wash., where he was born. He received his M.S. from the Montana School of Mines. From 1940-42 he worked as research assistant and instructor at M.I.T. He was with U. S. Economic Administration in Bolivia as production specialist from 1942-44. From 1944-46 he was with U. S. Army Corps of Engineers as First Lieutenant. At present time he is assistant professor of mineral dressing at M.I.T. An AIME member, Mr. Spedden presented a paper with Mr. A. M. Gaudin on "Flotation Microscopy of Cuban Manganese Ores" and "Magnetic Separation of Sulphide Minerals." Photography, canoeing, skiing and camping are his favorite pleasures.

J. C. Van de Water *(Castle Dome Overcomes Increased Haulage Grades, P. 951)* was born in Ellsworth, Kan. and attended the New Mexico School of Mines. He received his B.S. in 1929. He worked from 1929-1937 as engineer for St. Joseph Lead Co. From 1937-1943 he worked as assistant drilling and blasting foreman for Chile Exploration Co. and from 1943-45 as assistant mine superintendent at Castle Dome Copper Co. At present he is mine superintendent of Castle Dome Copper Co. Mr. Van de Water is interested in all phases of open-pit mining.

K. P. Wang *(Mineral Status of the Far East, P. 943)* was born in 1919 in Penhsihu, an important industrial center in Manchuria. Mr. Wang became acquainted with mining long before he chose this profession because his father, a prominent Chinese mining engineer, had been in charge of many large mines in Manchuria and North China. After graduating from the Chemistry Dept., Yenching University, Peiping in 1940, Mr. Wang came to this country to study at the Missouri School of Mines, where he received bachelor degrees in both mining and metallurgical engineering. In 1942 he worked as an engineer in the beneficiation plant of the Wah Chang Trading Corp., a well-known antimony and tungsten firm in New York City. His next job was with the Hudson Coal Co., where he had one year of training in anthracite mining and beneficiation at the company's Wilkes-Barre Collieries. In 1945 he returned to Columbia University to work for his doctorate degree. This was interrupted by a brief period of service in U. S. Marine Corps after which he became an American citizen; but he was able to complete the work for his Ph.D. at Columbia University in 1946. He is a member of the AIME, as well as of several honor societies such as Sigma Xi, Tau Beta Pi, and Phi Kappa Phi.

T. E. Wayland *(Use of Isopachous and Related Maps in the Florida Phosphate District, P. 975)* was born in Fountain City, Tenn. and attended the University of Tennessee. He received his A.B. in 1948. He spent two years (1949-50) with the geological survey on the Florida phosphate project. At the present time he is engaged in graduate study work at the University of Arizona, along with teaching under a teaching fellowship. Problems pertaining to economic geology are of particular interest to Mr. Wayland. His favorite past times are singing, bridge, fishing and mountain climbing.

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Drill and Carbide Grinder

Several refinements and improvements have been added to the Sterling drill and carbide grinder according to the manufacturer, *McDonough Mfg. Co.* One of these is a simplified adjustment to compensate for grinding wheel wear, and thereby improving the accuracy of setting the machine for various drill diameters. Another improvement is the permanent

mounting of a diamond wheel dresser on the drill grinding wheel guard. It is always available for easy dressing of the wheel and when not being used, swings back out of the way so that it does not interfere with the drill grinding operation. A light fixture mounted on a flexible tubing which allows positioning of the light at either end of the machine is another improvement that is now standard equipment. In addition to grinding standard drills, the manufacturer states that three lip core drills may be ground just as quickly and accurately on the Sterling grinder. **Circle No 1**

Spur-Gear Hoist

The Challenger, a spur-gear hoist incorporating new design innovations, is now in production in ½ and 1-ton capacities, according to the manufacturer, *Coffing Hoist Co.* The entire unit, including standard length of high-strength coil chain for an 8 ft lift, weighs only 39½ lb, thus is easily moved from one place to another, as needed. Strength and resistance to shock-load breakage are said to be gained by the use of formed steel plate in the housing in place of the more common cast aluminum alloy. The back plate is laminated to give extra rigidity for supporting the hoist mechanism. The Challenger is built so that it may be disassembled in a matter of minutes with simple tools. **Circle No. 2**

Mine Communication

Trolleyphones have been found to be particularly valuable in maintaining constant contact between dispatchers and locomotive operators whether on the move or stationary, at loading points or anywhere in the mine that instant contact is necessary for safety and highly efficient production. They can be used on anything that has a wire on it since they carry clear two-way talk over existing mine power circuits, ac or dc current. Using no signal wires, the trolleyphone operates by feeding FM carrier current into the mine power system. The heart of the unit is the transreceiver, containing the resistors and circuit—now mechanical in makeup by using ten convenient component parts units that plug into octal sockets like radio tubes. Featuring packaged maintenance the new design trolleyphone has its FM carrier circuit packaged in plug-in components for quick change by anyone in case of communication interruption. Available from *Farmers Engineering & Mfg. Co.* **Circle No. 3**

Shuttle Car

A new cable-reel crawler-type shuttle car for mine application has been announced by the Locomotive and Car Equipment Dept. of the *General Electric Co.* The new unit, designed for operation in close quarters, has a turning radius of from 14 to 16 ft. The new 250-volt car is driven by two sealed-type, 20 hp, 250-volt series-wound motors driving each track through single gear reduction, chain and sprocket. The cable reel is hydraulically driven with sealed-type enclosure. A spooling device controls level winding. The hydraulic motor torque is automatically compensated for reeling and unreeling cable pull. All internal control elements, including accelerating and headlight resistors are in sealed-type cases. Two steering levers at the operator's position operate the tracks. They are connected to the traction motor power circuits and to hydraulic brake valves. By moving a steering lever forward a few degrees, the corresponding traction motor is disconnected from the line, and the car turns slightly. **Circle No. 4**

Hoe Attachment

A newly designed hoe attachment for the Koehring 304 excavator will increase the machine's digging depth to 19 ft 9 in., according to a recent announcement by the manufacturer. Other improvements made on the model 304 provide extra resistance to side sway and extra strength to meet any operating condition for below ground level excavating. Position of the counter shaft in line with boom foot mounting on the 304 provides another important advantage. It eliminates dipper drift when the boom is raised. Use of double digging lines to the sturdy dipper eliminates reverse cable bends and results in important savings due to longer cable life. A versatile dipper arrangement provides top production capacity for the Koehring hoe under any working condition. There are no moving cables or sheaves on the dipper to clog with material and add to cable wear and breakage. Manufactured by *Koehring Co.* **Circle No. 5**

Floor Truck

Industrial Engineering & Mfg. Co., Inc. is now in production on a new line of floor trucks known as the Universal Stock Toter. This truck is a sturdy, highly maneuverable shallow bed truck designed for numerous jobs such as line stocking, machine stocking, moving work parts, or for con-

veying smaller parts in stockroom. The stock toter frame is of 1½x1½ in. x 3/16 angle iron. The container sides and bottom are of 12-gage sheet metal. It can also be furnished with sides and bottom of 13-15 gage ¾ mesh expanded metal. Caster equipment is two swivel-type at one end and two rigid at the other. The height of the truck from floor to top is 34⅛ in. with 4-in. casters.

Drilling Dust Exhauster

The Konigsborn drilling dust exhauster is particularly suited for roof bolting and in tunnel headings where multidrill Jumbos are used. It is provided with an adapter and can be used with any standard drill, sinker, or stopper. The following advantages are offered by the Konigsborn exhauster: dust-free air, obviating the use of dust masks and dust tight goggles; improved visibility, contributing to better and safer performance; strong air circulation, contributing indirectly to better ventilation and continuous air cooling of the bit, thereby increasing its life. The Konigsborn exhauster is manufactured in three models and is available through *Columbia Technical Corp.* **Circle No. 7**

Free Literature

(8) MINE HOISTS

Publication of a new 24-page bulletin which illustrates and describes various sizes and types of Nordberg mine hoists is announced by *Nordberg Mfg. Co.* These hoists are built in cylindrical, conical or cylindro-conical, single or double drum designs of the conventional or tandem type to meet specific hoistering requirements. Bulletin 190 shows with installation photographs the application Nordberg hoists have in coal and ore mining operations both in the United States and throughout the world. Descriptions of these installations give pertinent engineering data on the hoists' operation.

(9) DIESEL TRACTOR

The 32-page illustrated booklet describes practically every part of the *Caterpillar Company's* diesel D7 tractor with complete specifications. The engine, fuel system, and lubrication system are just a few features of the D7 tractor which are thoroughly discussed and illustrated. Some of the favorable points are: frame and steering clutch case built into a solid one-piece welded steel unit which provides for strength without excessive weight, individually removable slow speed steering clutches with long wearing metallic friction surfaces which allows for long operating life and accessibility. Another important feature which gives high tractor production without undue operator fatigue is the ease of steering of the D7. Only a light pull on the steering clutch lever is required to operate the steering clutch. The engine has large inspection doors on the right side of the crankcase and on both sides of the oil pan. These doors make it easy to inspect the main and connecting rod bearings and the oil pumps.

(10) VIBRATING SCREEN

A new 20-page illustrated book No. 2377 on model "UP" vibrating screens for accurate dry-screening of light and fine materials and model "NRM" liquid vibrating screens for the low-cost, high speed separation of solids from liquids, has been published by *Link-Belt Co.* Both types of screens are available in a wide range of sizes. The "UP" can be furnished with single or multiple decks, and with semi-enclosed or totally enclosed steel housings where required. Descriptive material includes specific information on how to select the right screen and screen cloth for maximum operating efficiency; dimension tables, weights, horsepower requirements.

(11) EARTHMOVING EQUIPMENT

Bulletin No. 895 issued by *Baker* on their bulldozers, gradebuilders and root rippers. The fingertip control provides for a direct connection or short linkage between control lever and control valve enables the operator to feel the minute movement of the cutting edge with fingertip ease. The frame and blade can be raised and hydraulically locked in an out-of-the-way position with positive hold to prevent blade from settling in order to permit maximum versatility in tractor operation without removing dozer. Design and location of the mounting brackets on either side of engine frame makes traction engine easily accessible for inspection and adjustment. These dozers are designed for the Allis-Chalmer tractor, model HD-20, HD-15, HD-9, HD-5 and are hydraulically and cable controlled.

(12) DIAMOND DRILL PRODUCTS

Available in all standard sizes, the complete line of Truco drill bits includes coring, blasting, pilot and casing bits. In addition, reaming shells are made in sizes to correspond with the various bit sizes. Truco impregnated core bits have been field tested in all sizes for use under the roughest hole conditions. Where surface set bits are susceptible to impact and abrasion from cherty, fractured or non-homogeneous formations, the Truco withstands the utmost in hard use. The concave blast hole bits are designed to be used in all softer formations where core samples are not required. An inherent characteristic of all solid bits is that the center has no cutting speed, a factor which tends to make the bit hard cutting and inclined to build up undesirable pressure Literature available from *Wheel Trueing Tool Co.*

(13) COAL CLEANING

A new catalog, recently issued by *Nelson L. Davis Co.*, describes the operation and gives detailed dimensions of their complete line of standardized heavy media coal cleaning plants. These plants, built in six sizes with capacities of 50 to 325 tons per hour will clean coal within any size range between ¼ in. and 10 in. Incorporated into these standardized units will be found all equipment necessary for precision coal cleaning such as is usually furnished for custom-built washeries.

(14) INDUCTION MOTORS

Construction features of large end shield bearing squirrel-cage induction motors are described in a new bulletin released by *Allis-Chalmers Co.* Built for a wide range of applications from central station auxiliary to general industrial drives, these motors are available in ratings and speeds up to 1750 hp at 1800 rpm. Construction features of these motors include welded stator yoke, long-life stator winding, capsule-type housings, protecting end shields and large discharge openings. The bulletin points out that the motors can be had with special electrical modifications to suit application requirements. Although sleeve bearings are standard, motors may be obtained with capsule-type anti-friction bearings whenever speed and application are suitable.

(15) PROTECTIVE HATS

To stimulate wearing of protective hats, *Mine Safety Appliances Co.* has published a new illustrated booklet which points out the importance of adequate head protection by listing eight representative examples from the many in MSA files where workers have escaped serious injury and even death when struck by falling objects while wearing Skullguards. A section of the booklet is devoted to the care of the MSA Skullguard. Proper precautions which prolong the life of the protective hat are shown. Important design features of a protective hat are also discussed in the booklet.

Mining Engineering **November**
29 West 39th St.
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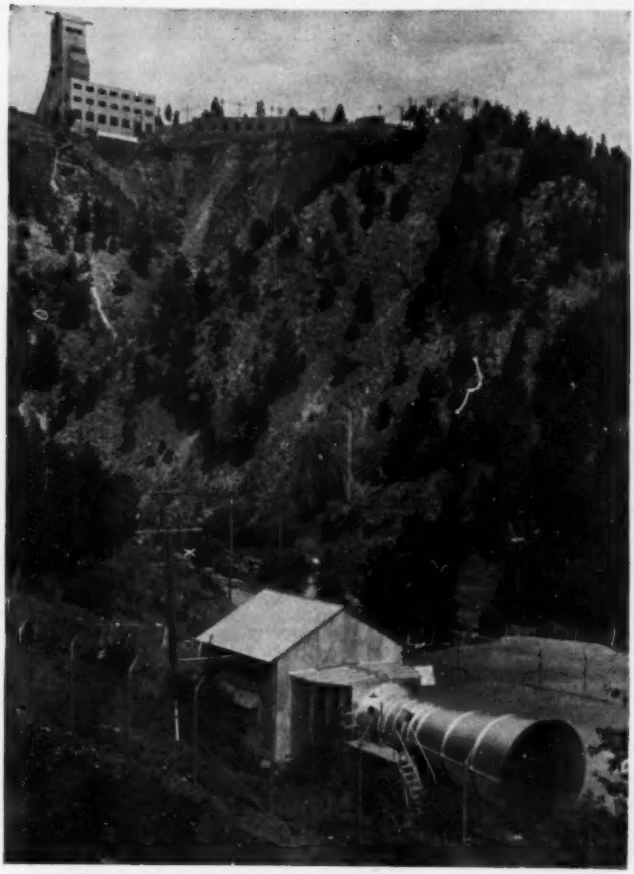


Trends

A DECISION to proceed immediately with construction of a 2½ million ton per year taconite beneficiating plant has been reached by Reserve Mining Co. The need for exploiting the taconites and the fact that they were ready for commercial development was pointed out in editorials in MINING ENGINEERING as early as September 1950 and again in July 1951. Thus Reserve, with this disclosure, has become the first to undertake such a project.

Present plans call for completion of the construction in 1955 with first operation late in that year or early in 1956. The new plant will be located on the north shore of Lake Superior about 55 miles east of Duluth at Beaver Bay. The plant will be connected by a 47 mile railroad with Reserve's mining property at Babbitt, Minn., on the eastern end of the Mesabi Range.

A $75 million construction contract has been awarded to a group of contractors. Included in this contract is construction of the railroad, the concentrating plant, a harbor and loading facilities, power generating and transmitting equipment, mining machinery at Babbitt, and two towns to house the thousands of men who will be employed. The contractors are the Hunkin-Conkey Construction Co., Cleveland, Ohio.; the Arundel Corp., Baltimore, Md.; and L. E. Dixon Co., San Gabriel, Calif. The three have joined under the trade name of Hunkin-Arundel-Dixon. Provision will be made in this construction program to enlarge the plant as required to a 10 million ton-per-year plant. Power plant capacity and heavy foundation work are included in the current construction project to accommodate at least part of this additional capacity.

It is estimated that more than 1½ billion tons of taconite averaging 23 to 24 pct iron is included in the Reserve Mining Co. holdings. This will produce 500 million tons of high-grade iron ore concentrate.

To operate the 2½ million ton plant at full production, the mining of 7½ million tons of taconite will be necessary. The ore will go through a crushing plant located near the pit which is being opened up at Babbitt and which will reduce the ore to −4 in.

The crude ore will then be transported by rail to the Beaver Bay plant where it will be further reduced to a powder about the fineness of cement. Concentration will be by magnetic separation after which the concentrate will be rolled into walnut size pellets and baked into hard balls in a pelletizing furnace. The finished pellets will be shipped to steel plants in the lower lakes region from docks within a new harbor adjacent to the plant at Beaver Bay.

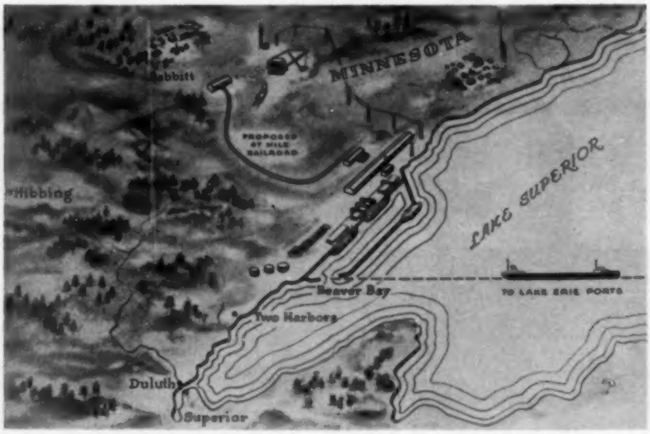
The result will be a tailor made product of much more uniform consistency than natural ore and which will be richer in iron units than any ore that has ever come out of the Lake Superior region. The process will take taconite with an iron content of about 24 to 25 pct and by beneficiation, will produce an iron ore pellet of about 64 pct iron. The cost of the pellets will be competitive with other Lake Superior ores.

Work is proceeding at the present time toward the re-equipping of a 300,000 ton-per-yr, one-section plant at Babbitt which will be in operation early next year. The Babbitt plant will permit the beneficiating process to be completely studied and developed to perfection while construction of the larger plant is under way. The larger plant will be designed in multiples of the process being developed in Babbitt. Eight such sections will be included in the first phase of the construction with 32 sections contemplated for the eventual 10 million ton plant.

Partners in Reserve Mining Co. are Republic Steel Corp. of Cleveland, Armco Steel Corp. of Middletown, Ohio, and National Steel Corp. of Pittsburgh. Manager of Reserve Mining Co. is Oglebay-Norton & Co.

Reserve is not alone in the taconite development program as both Erie Mining Co. and Oliver Iron Mining Co. have taconite developments under way. Erie, owned jointly by Bethlehem Steel Corp., Youngstown Sheet & Tube Co., Pickands Mather & Co., and Interlake Iron Corp., has been mining taconites and manufacturing pellets in a 2000-ton-per-day plant for nearly 3 years. Plans for a commercial-size plant, presumably 2½ million tons, have not yet been disclosed but are expected shortly. Oliver, the giant iron mining subsidiary of the U. S. Steel Corp., has constructed a fine-ore agglomerating plant at Virginia, Minn., and is tackling the toughest of the magnetic taconite metallurgical problems, that of agglomeration. Both sintering and nodulizing are being studied. Designed to produce about 1 million tons of egg-size nodules and sinter per year, the plant uses a 6x96-ft sintering machine and a 12x350-ft brick-lined kiln. Oliver will use the information gained from this plant when they begin large scale taconite operations.

PAUL Tyler's article "Russia's Mineral Potential" p. 494, June MINING ENGINEERING, has aroused discussion on a vital subject. Exemplifying this discussion is Norman Stines' criticism which is trenchantly stated on p. 949 of this issue. Mr. Stines is emphatic in his contempt for Russian technological know-how and



The north woods of Minnesota near the Canadian border will become an important source of iron ore for the American steel industry as the taconite deposits at the extreme end of the famed Mesabi Range near Babbitt, Minn. are soon to be developed. In the drawing, the principal features of the $75 million project are illustrated.

supports his contentions with anecdotes from his years of living in Russia before the revolution. In essence, Mr. Stines thinks that Russian claims of mineral output are grossly exaggerated. Some credence can be given this stand for it is well known that where Communism receives a serious setback, Soviet publications declare a great victory. With this experience as a guide, their more serious shortages may be looked for where the claims are most apparently exaggerated.

On the other hand mineral self-sufficiency has been claimed in recent weeks by A. N. Nesmeyanov, chairman since last year of the Soviet Academy of Sciences. He credited Soviet geologists with making discoveries so that nothing need be sought outside the vast territory dominated by the USSR. Until recently, when sanctions have been tightened against the Soviet, certain raw materials such as tin, tungsten, molybdenum, rubber, cotton and wool were imported. This indicates that they cannot be too strong in these materials.

Tyler, in his article, pointed up the discrepancy between mineral potential and actual production. In a territory as large as that occupied by the Communists, one must admit that there must be a great mineral potential regardless of whether available statistics are inadequate. To make minerals in the ground into finished metal products it takes machinery, skilled labor to operate it, and engineering. Herein lies the greatest obstacle to Russian self-sufficiency. Although encouragement can be gained from this and other Communist shortcomings, for the western world to dismiss Russian claims with complacency is dangerous.

ACCORDING to William E. Bullock, writing in the "Engineering Societies National News Letter" there is no shortage of engineers at all. He comes to this conclusion from the following considerations:

1—Leaders in the profession think that an "ideal composition" of the 350,000 engineers would be "one all-round, sound judgment, full-capacity engineer to every dozen specialists, technicians, detailists, mathematics manipulators, and similar assistant engineers."

2—There are only 60,000 of this type.

3—These assistants must be highly educated.

4—There are 90,000 non-college men coming up through the shop and field.

5—Engineering college enrollments are falling off and about 2000 engineers retire every year.

From these estimates he concludes that the solution is to re-educate as many of the 90,000 non-college men as possible. In concluding he states that "there is possibly no shortage of engineers whatever; there is simply an overlap between the inherited composition of the profession and the new requirements for cheaper precisely calculated solutions of problems which were formerly solved by expensive cut-and-try.

Some recently published facts about the engineering profession disclosed that about 75 pct of all engineers are employed in private industry, 7½ pct in government, and 2½ pct in education. Miners and metallurgists are 5 pct of the total, civils 25 pct, mechanicals 40 pct, electricals 20 pct, and chemicals 10 pct. There are about 65 workers for every engineer today as compared to 255 in 1900.

On the replacement side of the picture, the peak year of engineering graduates was 1950 when 52,000 received degrees. Estimates of graduates for the years 1951, '52, '53, and '54 are 38,000, 26,000, 20,000, and 17,000, respectively. The drop out rate from the time of enrollment until graduation among engineers is about 50 pct. Reliable sources estimate the replacement need for engineers to be about 20,000 engineers annually in normal production periods and 30,000 engineer replacements during a period of partial mobilization.

DUST control in drilling and removal of the cuttings is achieved by water in the United States and although dry dust collectors are used in conjunction with some roof bolting machines in coal mines here, the wet method is firmly entrenched and experiment is directed toward improving it. For instance some experiments are being conducted in the use of wetting agents to obtain better dust control. From Pauline Bryan of the British Information Service however, it is learned that considerable interest is being aroused in British mining circles by dry rock drilling with dust collectors. The principle is that the dust is extracted as formed by way of the core of the hollow drill steel, which is larger bore than used for wet drilling, and an extractor and collected in a cyclone separator and an air filter. This arrangement is receiving careful study by operators of deep mines in India and South Africa where at depths as great as 11,000 ft, the high temperature makes wet drilling undesirable because it increases the humidity of the atmosphere. The Dryductor drill, as the machine is called, was developed by Holman Bros. Ltd., of Cornwall, England, and is under trial there but is already in production for overseas export.

THE Congress recently appropriated an additional $425 thousand to complete the Leadville drainage tunnel. This will be used to cover the cost of extending the main tunnel beyond its present 9995 ft mark and driving laterals to other basins. It will also provide for retimbering, guniting in heavy ground, and concrete ditches where the floor is fractured.

Two of the major "underground reservoirs" already have been tapped, the Fryer and Carbonate Hill basins and a lateral is being driven to drain the Downtown basin in the area of the Penrose shaft. Tunnel crews working two shifts under the direction of Edward Matsen, Bureau of Mines engineer and Harry Greshuk, project manager for Utah Construction Co., contractor, have driven 200 ft in downtown lateral.

The water trapped in these basins by faults is being drained at the rate of 4.7 million gallons per 24 hours. As this water slowly drains, the workings that have produced $465 millions in lead, zinc, gold, silver and other metals are being exposed. The mines that were drowned when low metal prices made pumping prohibitive, will be able to resume operations and mine the still substantial ore reserves of the district.

Junction of the Leadville drainage tunnel and the 190 ft lateral to the Hayden shaft, which having been drained by the tunnel is now being rehabilitated by the Cadwell Mining Company. Water from other flooded Leadville mines flows through ditches under the long ties.

First actual production of titanium metal at the newly constructed Henderson, Nev., plant of Titanium Metals Corp. of America occurred in mid-October. Production will be expanded rapidly until the initial goal of 10-tons per day is reached. This will be equivalent to eight times the present world production of titanium metal.

F. Eberstady & Co., New York City, has announced plans for construction of a $400 million plant for the manufacture of synthetic liquid fuels from coal in southern Illinois, if the Government will guarantee a minimum price for the gasoline and other products to be produced.

The office of Price Stabilization authorized an increase of 2¢ per lb in the ceiling prices of domestic lead and zinc to encourage production for the mobilization program. OPS also set ceilings at 19¢ fob New York for imported lead and 19½¢ fob St. Louis for imported zinc.

A new type of low-cost, forged-steel rock bit which may be discarded after use, will be made at Midland, Pa., by a recently organized division of Mackintosh-Hemphill Co.

Appointment of William A. White, Sr., of Danbury, Conn., as director of the Miscellaneous Metals and Minerals Div. of NPA was recently announced. Harry B. Sharp, who has been acting director of the division, will remain as deputy director. Responsible for the flow of certain metals and minerals to meet the requirements of programs authorized by DPA, Mr. White will have under his jurisdiction more than 70 minerals, including asbestos, beryl, industrial diamonds, fluorspar, graphite, mercury, mica, monazite sand, selenium, and silver.

International Minerals & Chemical Co. expects to recover uranium as a byproduct at its new $10 to $12 million phosphate chemical plant under construction at Bonnie, Fla. The plant is scheduled for completion in late 1952 or early '53.

In a decisive move to break the log-jam of undelivered orders for steel copper, and aluminum now backed up on mill order boards, the NPA today ruled that any unfilled orders calling for third quarter 1951 delivery which are not shipped by Oct. 7 must be charged by a customer to his fourth quarter CMP allotment. Previously, an authorized CMP order accepted by a mill for delivery during the third quarter might be filled at any subsequent time and still be charged against a third quarter allotment.

Two special rack-rail electric locomotives which climb a 10 pct grade in Alpine winter weather are helping Austria's largest iron ore mine resume operation. The operation at Eisenerz requires the direct climbing of 10 pct grades. The 28-ton locomotives designed by General Electric use a series of 7 racks with a 10 pct maximum grade connected by rail sections having a maximum grade of 2.8 pct.

MINING ENGINEERING

EDITORIAL

LESSON IN IRAN

THE wave of nationalism commingled with communism which is sweeping from the Philippines across the Asiatic continent into the Middle East has climaxed in a tragedy in Iran which is shaking the foundations of the United Nations. Iran, who a few short years ago stood before the UN as a supplicant for protection against the occupation of the province of Azerbaijan by the Soviet, today defies its jurisdiction in the oil dispute with Great Britain.

The mining industry of the United States must watch this and other grave happenings throughout the world for whatever lessons may be learned for the administration of its far-flung investments. Is the keynote for the future to be the argument advanced by the Iranians that the natural resources of that sovereign state must be exploited for the benefit of the Iranian people to the exclusion of foreign operating control? Or will the patent claims of the British based on the legal premise of treaty and the fact that without them the oil would be undeveloped and useless to the Iranian people be upheld? Certainly history holds little hope for the latter eventuality.

Students of the East are disturbed by the success with which the Communists have tied successfully colonialism with capitalism by subtle propaganda. Communism maintains an active propaganda machine and the lethargy of the people toward the evils of that system plus a skepticism over the economic penetration by the United States creates a problem which must be combated by us as individuals, companies, and a Nation.

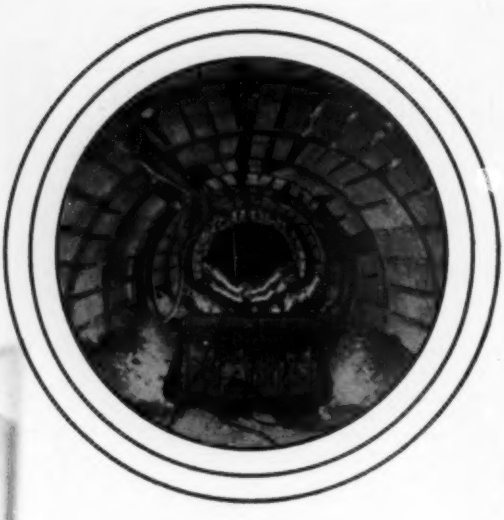
Astute diplomacy and the Voice of America are two mechanisms which may be used to combat these forces on a national level. Individuals must be careful of their comportment when working or traveling abroad and should not be afraid to staunchly support the democratic way in discussion groups.

More important are the methods of administration of American management of foreign enterprises. ARAMCO, the Arabian American Oil Co., operates undisturbed next door to the threatened area in Iran. The management of this concern may possibly serve as a pattern for successful operation in areas where unrest may be smoldering. Education of the natives and equal opportunity for employment by the company are an important part of company policy.

Our relations with Canada and Rhodesia which have encouraged the influx of American capital provide valuable experience. Canada has learned from association with the United States as we have learned from them.

Following these examples, a course presents itself to be followed by American companies who wish to invest abroad. Such investments should be regarded as 25-yr investments rather than 99-yr leases. This is sufficient time for the return of the capital investment, it opens up the mineral deposit for the benefit of the world and the country in which it is situated, and it allows time for training the natives. When the 25-yr period has expired the property can revert to native management with the American concern sharing on a royalty basis. This procedure could be a tangible answer to communist propaganda that the capitalism of this nation is linked with colonial imperialism.

The role of minerals in the course of world events has become starkly defined. With the Eastern and Western World racing for military supremacy, the minerals at stake in the Iranian controversy or in later upheavals could well be the measure which when thrown in the balance would precipitate a global war. These considerations make it imperative that the mining and petroleum industries take positive responsible steps to avert future incidents like Iran.



Circular Steel Sets Economical At Miami

by J. W. STILL

CIRCULAR steel sets have proved to be more economical for supporting slusher drifts in the block-caving mining used by the Miami Copper Co., at Miami, Ariz. The first steel sets were installed in July 1950, and up to July 1, 1951, about 474 sets (approximately 1896 ft of drift) were placed; the bulk of this work was done in the latter six months. All of the places selected for the steel were in heavy ground. Based on this experience the following definite conclusions can be drawn:

1—The steel sets are best suited and work well in highly altered rock, that tends to flow under weight and snugs in the whole structure (lagging, steel, etc.). However, where the rock pillar between the drift back and undercut breaks in large blocks and the weight focuses at individual points, steel sets are not suitable.

2—Installing steel in an advancing heading is somewhat simpler and easier than standing timber. Where steel is used for repairs, either in timber or steel lines, the repair procedure is not greatly different nor any more difficult than the old timber repair technique.

3—From a cost standpoint, the use of the steel sets to date shows a saving in normal maintenance cost, more than sufficient to justify its expanded use.

4—From a mining standpoint, an even larger cost benefit seems possible because steel sets permit more continuous and more uniform draw resulting in less dilution.

In the mining system used at Miami a substantial portion of the mining cost is stope timber, due to difficulties encountered in support of openings. Stope maintenance absorbed about 1/3 of all the timber used underground and about 1/6 of the mine labor crew. Neither concrete, steel, or timber will permanently support this weight in an active stope, and little forecasting can be done as to severity of weight to be encountered in any section. Experience at Miami has shown that, because of the weight problem, maintenance and drawing must be planned so that the entire stope will be drawn uniformly immediately after the undercut is completed. In heavy areas this is seldom possible. Those sections that take weight and fail generally require repeated repair resulting in non-uniform draw, high maintenance costs, less recovery, and greater dilution. This does not happen frequently, but often enough in the heavier areas to account for a large part of the overall maintenance cost. The importance of a uniform and regular draw cannot be overemphasized; for it is the only method of minimizing the weight and keeping it under control.

Faced with these mining conditions, studies were begun to determine a better method of supporting slusher drifts. A visit was made to the Mather "A" mine of the Cleveland-Cliffs Iron Co., Ishpeming, Mich., to observe the use of 13 lb, 4-in. steel H-beams in 9-ft lengths for posts and caps in support sets for main line haulage level (see MINING ENGINEERING, April 1950, "Development in the Use of Steel for Underground Support" by F. J. Haller). Experience at the Mather which was pertinent to the problem at Miami was that although the steel sets deformed under weight, they lasted longer than timber before complete failure.

"Rock Tunneling with Steel Supports" by Proctor and White and published by the Commercial Shearing and Stamping Co. of Youngstown, Ohio, provided valuable information. This book recommends the full circle set as best adapted to the worst weight conditions.

After further study, it appeared that experiments with full-circle steel sets seemed feasible. A trial order of 40 sets (enough for approximately 160 ft of drift) was placed and the steel was delivered late in June 1950. All of the steel used has been placed

MR. STILL is General Superintendent of the Miami Copper Co., Miami, Ariz., and is an AIME member.

in the slusher lines on the draw level and from July 1950 to July 1951 the following steel sets were placed:

New Headings	Repair Jobs Steel Replacing Timber	Repair Jobs Steel Replacing Steel
279	142	53

A total of 474 sets supporting 1896 ft of drift were placed.

So far steel has been used in places that were either known to be heavy; or in new headings where weight was anticipated.

The costs of steel vs. timber per ft of drift compares as follows:

Materials and Labor	Cost Per Ft
Timbered 93 bd ft timber plus $4 labor	$13.25
Steel Drift 29 bd ft timber, steel, plus $4 labor	18.50

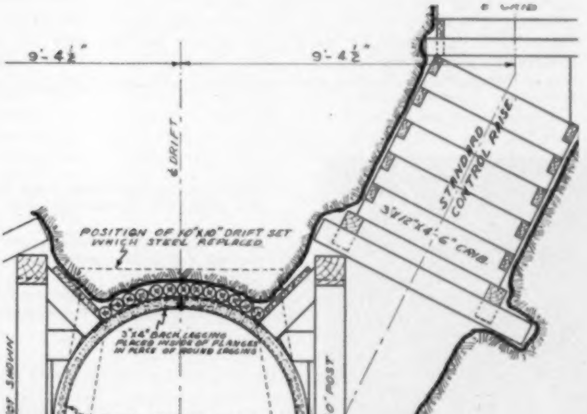
The above costs are computed basing timber at $100 per M at the working place (including framing shed and handling costs); and covers only the support material (timber or steel) and the labor of placing them. These are original installation costs but when general repairs are required these will vary from $40 to $55 per ft. With this cost comparison, anything that postpones or avoids a general repair job will obviously result in a sizable saving. When repeated minor repairs result in the failure of the steel itself and a general repair job is required, it is no more difficult or costly than for a timber repair job.

Observing the general behavior of the steel sets has shown that the weight apparently tends to shear around the 4 in. steel section (due to its small size) and break the lagging. As the lagging has only a 2 in. catch on the steel and a 3 ft in. span, this lets the ground move, bending and eventually breaking the lagging, with minimum pressure and damage to the steel set. In comparison, on the 10x10-in. timber drift sets (spaced 3 ft 9 in. centers) the lagging has a 5 in. catch and the relatively short lagging span plus the large exposed area of the 10x10-in. post and cap a major portion of the weight is placed on the set causing failure.

What the above adds up to in the excessively heavy areas is a continuing and sizable load of *minor repairs* in the steel as against *general repairs* in the timber. These minor repairs (changing side and back lagging) are relatively fast and cheap as compared to general repairs. In addition such minor repairs seldom shut a line down for more than a part of a shift, the timbermen are in and out of the line promptly, and ore drawing in the line is not seriously delayed. As against this, general repairs in the timber sections sometimes shut a line down for weeks at a time, with a resultant serious interruption to the drawing in the area.

Of the several applications where steel was installed only one was conclusively adverse. This was a half-line installation in 126 stope. Prior to steel installation the section had been over 50 pct drawn and had had several general timber repairs. In this part of the mine the rock is only slightly altered and the ground is consequently blocky and caves in large boulders. The steel sets in slusher drifts under such conditions are hard to relieve by back and side lagging changes, as the broken up pillar weight focuses in one or more large boulders. The steel quickly deforms under these conditions, the circular sets tending to go over on their sides forcing general repairs about as fast as would be required for timber under the same circumstances.

The heaviest and most costly steel stope was 114-S. The rock is highly altered, with a high moisture content and the stopes are small in area, low head, and small tonnage; all conditions which tend toward high maintenance cost and poor copper recovery. Stope 114-S was a duplicate of 114-N, where no steel was



A drawing of the steel sets installed at a draw raise, showing the conventional raise timber framed into a circular steel set. Steel sets are on 4 ft 5 in. centers at draw raises, 4 ft 1 in. in the drift between raises. Draw raises are spaced 16 ft 8 in. along the slusher drift.

used, and was a 5-line stope, 2 lines being timber and 3 being steel.

Both stopes are now completed, 114-N producing 165,323 tons and 114-S, 139,965 tons. The cost of drawing and maintenance combined of 114-S was 13.2¢ per ton cheaper than 114-N amounting to a saving of $18,475. The additional cost of using steel in 3 of the 5 lines totaled $1575 (300 ft at $5.25 per ft) which leaves a net apparent gain of $16,500.

Comparisons such as the above are not wholly conclusive, due to the numerous variables, however, all the pertinent conditions make these two stopes about as comparable as is ever possible.

Another installation that was extremely heavy is the E 564 line in 133 stope. This is the only steel line in 133 stope and was one of the first places where steel was installed as the original drift was driven. This area was heavy from the time it was undercut and the two parallel timbered lines 37½ and 75 ft to the east were partially down almost from the start. However, the steel line stayed open, with numerous minor repairs, until June; producing 37,281 tons in the 5 months since the undercut as compared to 32,011 tons from the adjacent timbered line in the same period. The maintenance cost in the steel line was about one third of the cost of the timbered line.

The following two installations produced results that were even better than hoped for. Stope 119 was one of the heaviest stopes ever encountered. Prior to the undercut, at the time the draw raises were about completed, and a start made on the undercut drifts, the entire stope (5 lines), took weight and all 5 lines closed in. One line at a time was repaired and undercut and the repairing and undercutting of the 5 lines in this manner lasted almost 5 months (an undercut area that in a normal stope would be cleaned up in 6 weeks). Subsequent to the undercut of the E 308 line, it was down and completely repaired about 3 times in 6 months; but despite this, little or no tonnage was pulled from the chutes in the center of the line. This E 308 line was again repaired, using steel in place of timber — 32 steel sets being placed. In the following 8 months, to the end of mining, the line produced 93,648 tons, and no general repairs were required as none of the steel sets failed. During this period the line continued heavy, requiring constant minor repairs, with timbermen in the line about 1 out of every 3 days. Only the fact that the steel held the line open and permitted a daily uniform draw kept the weight under control—a condition that had been almost impossible to attain with the timber sets.

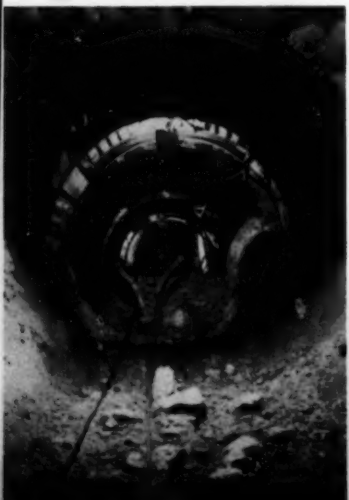
Another installation that is currently drawing and is acting in a similar manner are the two lines in 119-117 stope pillar, E 242 and E 276. These lines required complete repairs during December 1950 and January 1951, at which time steel was installed, 40 sets in each line. In the first 6 months of the year 101,572 tons were drawn from these two lines. It was necessary to change about 300 sets of back and side lagging but no steel set failures or general repairs occurred.

A 4-in. H-type steel section seems much too small for the ground support job at Miami. However, the full circle shape, the small 6½ ft diameter, and the approximate 4 ft center-to-center spacing gives a strong enough unit for the job. In addition, the relatively long (3 ft 9 in.) span of the lagging, which takes the weight and breaks, makes a unit where the main member, the steel, is somewhat protected. The circular steel sets have kept heavy ground open so that ore moved regularly with a minimum of delays and repair costs.

Circular type steel sets should be tied together so that the separate units will make a rigid structure for best results. Further, each individual set must be perfectly vertical, and so blocked or braced that it will stay in place until the weight snugs it in. The 3 in. angles used as collar braces, and the 4 in. H's or 3 in. angles used at the draw-raise positions make the structure rigid and permit its being placed in a vertical position.

Back and side lagging repairs should be made without delay. If delayed, the steel may take a minor deformation which weakens it for later weight.

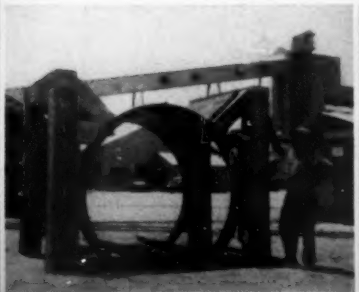
So far no attempt has been made to salvage any of the steel sets at the end of mining. The sections abandoned to date were in such condition that cost and safety considerations made this advisable. However, of the steel salvaged from repair, some 118 steel sections (or 39 1/3 full sets) have been straightened and will be re-used.



At left, a slusher drift equipped with circular steel sets that proved more economical than timber and permitted even draw without shutdowns.

In center, yard setup of two sets with timber for a draw raise. Rails on bottom for slushing. This is shown in drawing on preceding page.

At right, lagging takes weight and fails, relieving steel set. Lagging on the right side of set is new. The failure of the lagging reduces the deformation of steel.



Mineral Status of the Far East

The mineral potential of the Far East, important to the United States for tin and tungsten as well as other minerals, is set forth in this first installment of a two part article by a specialist in the Far Eastern field. In this installment the mineral production and resource picture of China, Japan, Korea, and Formosa are discussed. The second installment, in December, will cover Hong Kong, Indo-China, Thailand, Burma, Malaya, Indonesia, the Philippines, and British Borneo.

by KUNG-PING WANG

THE Far East is again in the limelight of world affairs. In fact, it has been the cradle of conflict for more than 50 years. Several centuries ago while a number of western European nations were competing to establish colonial empires in southern and southeastern Asia, few noticed that Russia was quietly detaching from China large areas in Siberia. In 1842 the British took Hong Kong, paving the way to extraterritorial rights in China. Soon after, (1853) Commodore Perry knocked at the gates of Tokyo, revealing the industrial and military might of western civilization and Japan convinced itself of its destiny to share in the spoils of Asia. Meanwhile, Russia in search of a warm-water port penetrated Manchuria by building the Sino-Eastern Railway from Siberia to Port Arthur and Dairen. This was a blow to Japan, which, after subjugating Korea, was looking toward Manchuria and its rich resources as a mainland base for further expansion. The result was the Russian-Japanese War of 1905, which Japan won while the western countries were busy consolidating their positions farther south. China and Japan were allies during World War I, but Japan forcibly took over the extraterritorial concessions of Germany. In 1931 Japan annexed Manchuria outright on the correct assumption that world opinion would be divided on the issue of aggression and that no action would be taken. The so-called *Manchurian Incident*, based upon the famous Tanaka Memorial of world domination, thus laid the groundwork for World War II. Meanwhile Russia, having been temporarily ousted from Manchuria, quickly established its control in Mongolia as well as the fringe areas of Sinkiang. An opportunity to resume economic penetration of Manchuria came again during the Yalta Conference in 1943. Immediately after VJ-day the looting of Manchuria indicated that Russia was not at all sure that the Nationalist Government of China would either go along with the agreement made at the Yalta Conference or could be overthrown on the mainland, but its incessant efforts to foster the Chinese Communist regime soon paid dividends because of Chiang's mistakes and the indecision of the West. Recent events in Korea indicate the strong interests of the Soviet Union and Communist China in the resources and strategic position of that little country. Undercover forces of nationalism cleverly exploited and masterminded by communist powers are rumbling in Asia. The Chinese Communist regime has already committed itself to the complete "liberation" of Korea, the active support of the Viet-Minh regime in Indo-China, and the invasion of Formosa. Moreover, it is appealing to some 11 million overseas Chinese, mostly in southeastern Asia, to endorse its aggressive actions. The traditional concern of the Soviet Union over its Asiatic position, the stirring of awakened though misguided masses of China, the population pressure of Japan, the reliance of the West on certain raw materials in the area, and the strategic concepts of many nations, make it obvious that the Far East will remain one of the focal points of collision of world interests for some time to come.

Actual mineral development in the Far East, except in Japan, is far below western standards. In fact, relatively little of the resources have been explored, but available data indicate that the Far East is rich in mineral commodities of world significance. The area can produce the bulk of the world requirements of tungsten, tin, antimony, graphite, and talc and has exportable surpluses of manganese, chrome, bauxite, phosphates, mica, quartz crystals, mineral sands, and possibly mercury and molybdenum. It has large potential resources of iron ore, coal, and chemical and construction raw materials. If the petroleum production were evenly distributed, the area would be also self-sufficient at the present consumption level. The apparent deficiency in nonferrous base metals can probably be overcome by intensive exploration, inasmuch as there seems to be a rich mineral belt that extends from southwestern China to northern Burma, Indo-China, and Thailand. In general, the area, although not as rich as the Western Hemisphere, has the mineral resources to permit moderately expanded industrial economy, but the creation of sufficient processing and fabrication facilities must await political stability and a higher purchasing power. So far, exploitation has been confined largely to minerals and metals that either have export markets or those, such as fuel and construction raw materials, that could easily be processed for domestic consumption. Both the Soviet and Western blocs depend on the area for certain export items, but the USSR by its close ties to Communist China has already overcome its major mineral deficiencies.

The countries included in this report are China (excluding Formosa), Japan, Korea, Formosa, Hong Kong, Indo-China, Thailand, Burma, Malaya, Indonesia, the Philippines, and British Borneo. Unless otherwise noted, the metric ton is used as the unit of measure for the following countries: China, Japan, Korea, Formosa, Hong Kong, Indo-China, Thailand, Indonesia, and the Philippines. The long ton is used

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Estimated Mineral Resources and Production in the Far East[1]
Selected Commodities

Commodity	Unit	Reserves: Quantity	Reserves: Far East Countries	Mine or Metal Production[2]: Quantity	Mine or Metal Production[2]: Percent of the 1943 World Output	Mine or Metal Production[2]: Main Far East Producers
Coal	Million M.T., all types	290,000	C—86%, Ic—7%, J—6%	130	7.1	C—46%, J—39%, K—7%, Ic—2%, F—2%
Crude petroleum[3]	Million barrels	1,600	In—65%, Bb—31%	88	3.9	In—57%, Bb—35%, B—6%
Refined petroleum	Million barrels	—	—	76	3.4	In—71%, Bb—19%, B—6%, J—3%
Coking coal	Million M.T.	10,000	C—90%, J—10%	18	7.2	J—55%,[4] C—45%
Iron ore	Million M.T., all grades	8,000	C—67%, Ph—13%, In—12%	16	7.0	C—69%, Ph—12%, M—12%
Pig iron	Million M.T.	—	—	6.9	6.0	J—61%, C—29%, K—8%
Crude steel	Million M.T.	—	—	9.3	5.3	J—87%, C—9%
Rolled steel	Million M.T.	—	—	5.5	n a	J—90%, C—10%
Manganese ore	Thousand M.T., all grades	35,000	C—86%, Bb—3%, Ph—2%	610	15.0	J—66%, C—12%, Ph—8%, In—8%, K—5%, M—1%
Chromite	Thousand M.T., all grades	15,000	Ph—72%, Ic—13%, J—5%	410	22.8	Ph—80%, J—19%, Ic—1%
Tungsten ore	Thousand M.T., 65% WO_3 ore	6,000	C—58%, K—20%, B, T.	33	58.5	C—43%, B—24%, K—24%, T—5%, J—3%
Aluminum	Thousand M.T., metal	—	—	150	7.7	J—76%, F—9%, K—8%, C—5%
Copper	Thousand M.T., copper-in-ore	3,500	C—57%, J—31%, Ph—5%	132	4.9	J—71%, Ph—8%, C—7%, K—5%, F—5%, B—3%
Lead	Thousand M.T., lead-in-ore	2,000	C—50%, B—25%, J—11%	136	8.7[5]	B—56%, J—16%, K—15%, C—9%, H—4%
Zinc	Thousand M.T., zinc-in-ore	3,000	J—47%, C—33%, B—17%	180	10.0[5]	J—52%, B—19%, Ic—11%, K—11%, C—7%
Tin	Thousand M.T., tin-in-ore	4,000	M—35%, C—35%, In—20%	175	119.3	M—49%, In—26%, T—10%, Ch—9%, B—5%, Ic—1%
Antimony	Thousand M.T., antimony-in-ore	4,500	C—89%, Ic, Bb, T.	44	82.9	C—90%, Ic—3%, B—2%, Bb—2%, T—1%, J—1%
Asbestos	Thousand M.T., all types	2,000	C—65%, J, K.	28	4.4	J—47%, C—34%, K—17%, Ic—1%
Mica	Thousand M.T., all types	n a	C, K.	5	7.1	C—95%, K—5%
Graphite	Thousand M.T., all types	n a	K, J, C.	139	49.6	K—75%, C—16%, J—9%
Fluorite	Thousand M.T., contained fluorite	4,500	C—48%, K—48%, J—2%	235	22.8	K—55%, C—42%, J—2%
Talc and pyrophyllite[6]	Thousand M.T.	75,000	J—93%, C—3%, K—3%, Ic, T.	560	70.8	J—71%, C—21%, K—7%
Magnesite	Thousand M.T.	5,000,000	C, K.	850	27.0	C—81%, K—19%
Bauxite	Thousand M.T.	n a	In, Ic, M,Pl, C.	968	6.8	In—67%, M—18%, Pl—14%, Ic—1%
Phosphates	Thousand M.T., all types and grades	150,000	Ic, C, Chr, D, Pl.	847	9.5	Chr—29%, D—24%, Pl—17%, C—12%, Ic—12%, K—4%, In—2%
Pyrites[7]	Thousand M.T., sulphur content	160,000	J—63%, C—32%	1,120	23.0	J—90%, K—5%, C—3%, F—2%

[1] Symbols for countries as follows: C (China), J (Japan), K (Korea), F (Formosa), H (Hong Kong), Ic (Indo-China), T (Thailand), Ph Philippines), In (Indonesia), B (Burma,), M (Malaya), Bb (British Borneo), D (Daito Island, formerly Japanese), Chr (Christmas Island), and Pl (Palau Island).
[2] Sum of peak outputs of all countries for different years.
[3] Chinese reserves of 4.4 billion bbls not included since most of it is oil from shale.
[4] Includes semicoking and noncoking coal, which could be blended with coking coal to make coke.
[5] Based on world metal rather than mine output.
[6] Japanese and Korean reserves and production almost entirely pyrophyllite.
[7] Japan also produced a peak of 230,000 tons of native sulphur, or nearly 6% of the 1943 world output.

for Burma, Malaya, and British Borneo. A table on the mineral resources and production in the Far East for selected commodities is included to show the potentialities of the area in relation to the world picture. The sums of the peak outputs for various countries are used in the column on production because of the sporadic nature of mineral exploitation in the Far East. The world production of minerals in 1943 is arbitrarily chosen as a basis of comparison.

China (excluding Formosa)

China's mineral potentialities stand high among the world nations. But in terms of per capita share its position is not outstanding. Furthermore, mineral production is small and has contributed little toward the economy of China. The major exception is coal, which has provided the limited number of industrial plants in the country with electricity and power and the small railroad network with fuel. The production of coking coal has been only a fraction of the total coal produced because the iron and steel industry is virtually undeveloped. Also helpful to the Chinese economy has been the valuable foreign exchange earned from the export of tungsten, antimony, tin, and mercury. In general, political instability in the last few decades has not favored rapid mineral development; the Manchurian industry, however, was built up by the Japanese only to be disorganized again by the Russians. China cannot make much progress in the future unless substantial foreign assistance is forthcoming.

Coal reserves are said to be some 250 billion metric tons, placing China fourth among the world nations. Although the peak annual production has been less than 60 million tons, coal has contributed much to whatever modernization China can claim. The present production rate of perhaps 45 million tons can be greatly increased in the future. About half of the Chinese coal is now produced in Manchuria, while most of the rest is from North China. The distribution of the country's reserves are such that South China has only about 5 pct and the famous Shansi-Shensi Basin, with 80 pct, has hardly been touched because of lack of transportation and markets. It is also significant that most Chinese coals are relatively high in ash and the coking coals, though abundant, are not always near iron-ore deposits.

The potentialities of water power are also great. Possibly over 30 million kw could be developed

from China's famous rivers, notably the Yangtze, the Huangho, Sikiang, Sungari, Yalu, and their tributaries. It is claimed that 10.5 million kw can be developed in the Upper Yangtze alone. However, most of the resources actually developed are in Manchuria, where the remaining Japanese-installed capacity is perhaps 400,000 kw, which, although small, has been a vital factor in the rehabilitation of the region. The known petroleum reserves are small in comparison with coal and water power, but the country has not been carefully explored. These reserves are said to be 4.4 billion bbl, of which half is credited to Manchurian oil shale and nearly 20 pct to Shensi shale. Kansu and Sinkiang resources, however, are not included in the above estimate; they show great promise, despite the fact that they are not easily accessible to coastal consuming provinces. Present oil production, mostly from Manchuria and Kansu, is less than 1 million bbl per year, far short of the country's requirements, especially since the beginning of the Korean War.

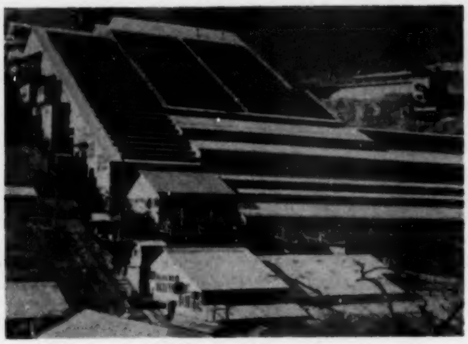
Next to India, China has the largest iron resources in the Far East, with some 5.5 billion tons of ore reserves. However, because 95 pct of the 4.7 billion tons credited to Manchuria is low grade, there remains only about 1 billion tons of economically workable ore. This is nevertheless sufficient for building a moderately large iron and steel industry, which at present is in the infant stage. Iron and coal resources are such that major centers can be established only in the northern coastal part of the country. So far, southern Manchuria is the only area of any significance, and its 1950 production of about 720,000 tons of pig iron, 460,000 tons of crude steel, and 300,000 tons of rolled steel represent over 70 pct of the national output. China definitely is deficient in finished iron and steel products, a situation that is particularly serious on account of the absence of substantial imports. Production undoubtedly will increase somewhat, but it will be limited by the availability of equipment to replace that removed from Manchuria by the Soviet Union. China would be doing well if it could attain in the next 5 years the peak production level attained under Japanese occupation. The possibility of resuming iron-ore shipments to Japan in exchange for finished products should not be ignored.

China's position in ferroalloy minerals is strong in certain respects and weak in others. It has large resources of manganese and tungsten, sufficient molybdenum and vanadium, but little chromium, nickel, and cobalt. Moreover, none of these minerals are consumed domestically to any great extent, although tungsten has been shipped all over the world, molybdenum formerly to Japan and now to Russia, and both manganese and vanadium formerly to Japan. Manganese-ore reserves are of the order of 30 million tons, of which two thirds can be considered high grade. Production, mainly for export, reached a peak of nearly 80,000 tons in 1937. China's tungsten reserves, particularly in southern Kiansi, are fabulous, amounting to 2 to 4 million tons of concentrates. The 1950 output probably came to about 12,000 tons of concentrates, completely solving the shortage of tungsten in the Soviet Union. This output can, moreover, be increased to 15,000 tons without much difficulty. Tungsten and tin are the two main dollar earners among China's minerals, having played an important part in obtaining loans from the United States during the war and in the $300 million barter agreement between the Communist regime and the Soviet Union. Although molybdenum is found in many parts of the tin-tungsten regions of southern China, the major producing mine is Yang-chia-chang-tzu in southern Manchuria; reserves are about 20,000 tons of contained molybdenum, and maximum annual production was about 1000 tons of 75 pct MoS_2 concentrates. The Soviet Union, being deficient in molybdenum, presumably has assisted the Communist regime in reopening this mine. Several important iron-titanium-vanadium deposits were discovered by the Japanese in the Luanp'ng and Ch'engteh districts in Jehol, where they were exploited for the manufacture of ferro-vanadium in a plant near Chinchow that had a yearly capacity of 80 tons (35 pct V).

China has adequate nonferrous base-metal resources to permit a moderate program of development, but production has hitherto been far below domestic requirements, particularly in terms of finished products. Copper reserves are of the order of 2 to 3 million tons of metal. Most of the high-grade ore occurs in southwestern China, principally in the Tungchuan district, Yunnan Province, where production can eventually attain 10,000 tons per year; lower-grade ore has been exploited to a greater extent in recent years at the Shih-tsui-tzu, Fu-Jung, Hua-t'ung, and other mines in southern Manchuria. The 1950 output of refined metal in China, mostly Manchuria, possibly reached 5000 tons, but mine output did not exceed 3000 tons. Over a million tons of copper has been taken from Tungchuan alone, explaining a reserve of secondary copper. Lead and zinc reserves may each be about 1 million tons. The only major mine in China proper has been Shui-kou-shan, Hunan, which, although once producing some 10,000 tons each of lead metal and zinc in concentrates yearly, cannot be expected to regain its past prominence. However, the Huili district, Sikang, has great possibilities. Five mines in Manchuria, namely Ch'ing-cheng-tzu, Yang-chia-chang-tzu, Huan-jen, Hsui-yen, and T'ien-pao-shan, together exceeded the Shui-kou-shan peak, but they are at present producing at about one third of their maximum rate.

China has long been the principal world source of antimony. Its output from 1929 to 1937 ranged from 14,000 to 20,000 tons per year and represented 37 to 76 pct of the world total. Present production

The new coal preparation plant at the No. 2 Mine of the Ashibetsu Colliery in Hokkaido, Japan.



Mikobata Concentrator in Hyogo, Japan, milling ore from the Akenobe Mine. The mill makes concentrates of copper, tin, lead, and zinc. Ore is hand sorted at the mine.

Living quarters at the Kawayama Mine, Japan. The mill handling cupriferous-pyrrhotite ore has a capacity of 7500 metric tons per month, and plans are to increase this to 20,000 metric tons.

rate may be only 5000 tons, because it is limited by the domestic and Soviet demands. Antimony reserves, however, are extremely large, totaling 3.5 to 4 million tons of metal. The most important mine is Hsi-kwang-shan, Hunan, which normally produces more than half of China's output. China, with a normal production of 10,000 to 15,000 tons per year, usually ranks fifth as a world producer of tin. The 1950 output of some 7000 tons can easily be increased to 10,000 tons, provided more 99.6 pct refined tin can be produced to satisfy Soviet specifications. The major tin district is Ku-chiu, Yunnan; two lesser districts are the Fu-ho-chung area, Kwangsi, and the southern Kiangsi tungsten areas. Reserves of the country are about 1.5 million tons, and Ku-chiu alone can, with modern methods, furnish at least 20,000 tons yearly. Several hundred tons of mercury can be produced annually in Kweichow Province alone; the Soviet Union imported more than 600 tons from China during World War II. The country has also produced over 200 tons of bismuth concentrates per year as well as several thousand tons of arsenic, both by-products of tin-tungsten mining. As a source of aluminum, the country has a 271-million-ton bauxite-ore deposit in Poshan-Tzuchuan, Shantung; a 156-million-ton boehmite deposit near Kunming, Yunnan; and a 351-million-ton alunite deposit in Kansu. Gold is also important in China, and uranium ores occur in Hai-ch'eng, Manchuria, Papu, Kwangsi, and Sinkiang.

China has relatively large resources of most of the major nonmetallic minerals. For the chemical industry, pyrite, gypsum, and nitrates are sufficient; soda and salt, including brine, rock, and lake salt in the interior provinces, are far above China's requirements. The little-known phosphate deposits are also large. Japan took almost 100,000 tons of fluorite from China yearly. Magnesite deposits in Manchuria may prove to be the largest in the world, and good metallurgical limestone, fireclay, and kaolin are abundant throughout China. Even minerals like talc, graphite, asbestos, and mica are sufficient. Briefly, it can be said that resources of nonmetallic minerals are substantial, but production, except for the items that a backward economy can use and those that Japan needed, has been small.

Japan

Japan is by far the most highly industrialized country in the Far East, comparing favorably with many European nations. The disintegration of the Japanese Empire has not impaired its basic structure, and the country will rise again if adequate raw materials can be made available. At present, Japan is handicapped by the greatly reduced trade with Communist China and war-torn Korea, but other trade channels, such as southeastern Asia, have been re-established to a great extent. Postwar recovery has been rapid under U. S. occupation, especially since the beginning of the Korean War; the rate of industrial production in 1950 has already exceeded that of 1932-36. The mineral and power industries have played an important part in putting Japan back on its feet.

Among Japan's assets, water power is outstanding. Potential resources are said to be 20.4 million kw, of which 6.6 million kw have already been developed. The 1950 hydro-energy output of some 43 billion kw-hr established a new record and constituted about 90 pct of the total electric power generated. Aware of this naturally rich resource, the Japanese are now seeking capital for further development. The coal reserves in Japan, chiefly bituminous, are of the order of 16.7 billion tons, but the deposits are generally low grade, noncoking, and costly to mine. As a fuel, however, coal is indispensable, and the 1950 output has already been increased to 39.7 million metric tons, only about 12 million tons below the wartime peak. Increased mechanization of mining, washing, and blending have become necessary to counteract low productivity and more rigid market specifications. Petroleum resources are small, less than 4 million kiloliters of known reserves, but prospecting and production have received much attention; as a result, domestic crude output reached nearly 1000 kl daily in the latter part of 1950. However, the bulk of Japan's crude requirements must still come from abroad; 1.34 million kl was imported in 1950. Refining capacity in the country, on the other hand, is about 8000 tons a day, and the Pacific coast refineries have been reopened to process imports.

Japan's iron and steel industry is also well established; its theoretical annual capacities are 5.8 million tons of pig iron, 11.6 million tons of crude steel, and 8.7 million tons of finished rolled steel. Actual outputs in 1950, however, were only 2.2, 4.8, and 3.4 million tons, respectively, postwar records. High cost of coking coal and the inadequacy of iron materials are the basic obstacles to higher production. Most of the coking coal formerly came from China, a channel that is almost completely blocked,

and costly coals from the United States and elsewhere have been imported. To reduce imports of coking coals, the Japanese are investigating the production of coalite (A char product of low-temperature carbonization that could be blended with other domestic coals for coke manufacture) from domestic semicoking coal. The iron-material supply problem is even more critical because scrap stocks are diminishing and present domestic ore production cannot greatly exceed 1 million tons annually. Again, China's supply, totaling more than 2 million tons of contained iron in 1942, has been cut off, and the present receipts from normal channels, as Malaya, Philippines, Hong Kong, and other areas, will be less than 1 million tons of contained iron in 1951. Although the large reserves of domestic iron sands may eventually be utilized, Indian iron ore may be made available, and greater production may be promoted in southeastern Asia; the main problem still is China. The ferroalloy industry is also well advanced, but raw materials, except chromite, which is adequate, and manganese, which can be obtained at high cost, are seriously deficient.

The peak production of refined copper in Japan was 122,849 tons in 1943, of which domestic ores contributed 94,729 tons; the 1950 figures are 84,749 and 39,322 tons respectively. In the latter year, however, the difference is made up from stocks rather than from imports. Although Japan has been exporting large quantities of copper to the United States, domestic consumption is increasing while the production of refined metal is declining and stocks are being depleted. Soon Japan must try to find import channels and/or increase its domestic mine production of high-cost copper from reserves that are estimated to contain 1.1 million tons of copper.

The zinc situation is more favorable than the copper situation, since zinc ores are much higher-grade. Productive capacities, however, are lower; the 1950 outputs of 49,024 tons of refined metal and 52,032 tons of contained zinc in domestic ores were, respectively, 72 and 58 pct of the 1943 peak. Japan can retain its self-sufficiency in zinc for some time. Zinc reserves are estimated at 1.4 million tons of metal, and productive capacity for refined metal is 74,000 tons a year. Future demand probably will not greatly exceed 50,000 tons a year. Secondary-zinc recovery can also be explored, and, barring a major world crisis, the country may also become an exporter of zinc. Lead is the weakest link in the nonferrous base metals. It occurs with zinc in a ratio of 1:5, and the reserves are only 260,000 tons. Secondary lead, however, is recovered; the present smelter capacity is 26,000 tons a year, and the refinery capacity, 45,000 tons. The 1950 outputs of lead in concentrates and refined lead were, respectively, only 10,853 and 16,036 tons, as compared with peaks of 21,207 and 34,929 tons in 1943. Postwar exportable surplus, mainly to the United States, is a temporary phenomenon, as potential demands far exceed productive capacities.

The aluminum industry, with a capacity of over 50,000 tons a year, depends entirely upon imports for raw materials. Bauxite from Indonesia and Malaya made possible a production of 24,764 tons of aluminum in 1950; it is planned to increase the output to about 35,000 tons in 1951. The Japanese have been interested in redeveloping the deposits in Palau Island, which they mined during the war, so that the excess Japanese capacity could be fully utilized. The magnesium industry is now idle, although the country once produced nearly 3000 tons annually. Japan produces small amounts of antimony, mercury, tin, cadmium, gold, silver, and platinum, but the country in general is seriously deficient in reserves and production. There is nevertheless adequate bismuth and arsenic.

Of the chemical raw materials, pyrites and sulphur are more than adequate. The 1950 output of refined sulphur reached 84,836 tons; annual production probably could be increased to perhaps 200,000 tons, leaving a small quantity available for export. The 1950 output of pyrites was 1.9 million tons, which approached the peak level. Pyrites and other sulphides are supporting a large sulphuric-acid industry. Pyrite sinter has also been used as a source of iron. Japan has a 5-million-ton cement industry, and its gypsum and limestone (including metallurgical type) resources are also large. However, other nonmetallics, such as phosphates, magnesite, fluorspar, talc, graphite, and mica, are seriously deficient, while salt production is not large enough to supply the country's needs.

Korea

Korea has attained as high a technological level as Manchuria, although its resources are, except for certain items, generally smaller. Like Manchuria, its industry was at one time oriented toward the Japanese economy. It is known that Russia removed little equipment, but since the Korean War major demolition of industrial plants which are mainly in North Korea, has paralyzed the country's mineral production. Details regarding the present status are not known. Nevertheless, a review of past performances will indicate the future potentialities of the country. Most of the mineral and power resources are north of the 38th parallel in an area that is intimately related to the economy, geology, and industry of southern Manchuria. The only worthwhile minerals in South Korea, which is the main agricultural base of the country, are tungsten and, to a lesser extent, graphite and molybdenite.

Korea is rich in water-power resources, with a potential of 6 million kw and a capacity before the present war of 2.5 million, virtually all in North Korea near the Manchurian border. This power supply has been important not only to Korea but also to Manchuria. In general, however, developed hydroelectric power is inadequate to meet the Nation's requirements, and the fuel resources of the country are meager. The over-all power situation is thus not promising. Coal resources, chiefly anthracite, may be half a billion tons, and normal annual production has been close to 6 million tons. Again, this source of fuel is largely in the north; the anthracite mined in South Korea is low grade, containing as much ash as coal. Oil resources are lacking in the country, but synthetic liquid fuels formerly were produced in Korea. A refinery at Wonsan, now demolished, processed Sakhalin, Indonesia, and Borneo oil.

There are about half a billion metric tons of medium-grade iron ore in the country, while normal production under Japanese occupation was 2 to 3 million tons of concentrates. The main mine center is Musan, near Vladivostok. Pig-iron and coke annual capacities were about 1 million tons each in 1944, but coking coal must be imported. Steel and rolling-mill capacities were much smaller than pig capacity. Most of the iron and steel and ferroalloy

The Kasei Iron Mine in Korea. Note the large number of workers in the pit.

plants near Pyongyang, Chongjin, and Songjin presumably have been destroyed recently. The largest tungsten mine in the world is at Sangdong, South Korea, and the country probably produced over 5000 tons of concentrates annually late in World War II. Nearly 500 tons of molybdenum in concentrates and 1000 tons of manganese ore were also produced yearly. Korea rivaled China in fluorite production, with over 130,000 tons in 1944. Except for coking coal, Korea has sufficient resources to maintain a small iron and steel industry.

Copper resources are not large, but lead and zinc reserves are substantial. Respective maximum yearly mine outputs were 7000, 20,000, and 20,000 tons of metal, while smelting capacities, except for copper, were slightly lower than mine output. Aluminum and magnesium capacities were about 30,000 tons and 7000 tons a year. At one time over 1 million ounces of gold was produced annually. The main processing centers for nonferrous metals are near Pyongyang, Yongampo, Munpyong, and Hungnam, several of which may have been destroyed in recent fighting. It is apparent that Korean nonferrous resources will permit moderate outputs, and cheap power makes it economical to exploit them.

Korean graphite is well known in world markets; the 1944 output was over 100,000 tons, of which one fourth was the crystalline variety. The main crucible manufacturing center is Chinnampo. Korea also has produced enough mica, asbestos, soapstone, apatite, pyrite, and salt for its needs. Magnetite resources are among the world's largest, and good-grade limestone and clays are abundant. During World War II the country produced 1.2 million tons of cement annually.

Formosa

Formosa is rich in agricultural production but relatively poor in mineral resources. Its small mineral and metal industries, formerly geared to the Japanese economy, have been greatly reduced in capacity as a result of wartime bombings; they are slowly being rehabilitated to utilize local raw materials for manufacturing the products needed for domestic consumption. The island has no outstanding mineral product, and it appears to be deficient in a number of essential resources, such as petroleum; iron, and fertilizers. Most of the major mineral enterprises are now controlled by the Taiwan Industrial & Mining Syndicate under the National Resources Commission. So far, ECA assistance in the mineral field has been small, the help rendered being mainly in goods rather than in development.

Coal resources on the island, largely on the northern tip, are said to be only about 400 million tons. Much of the coal occurs in thin seams that are costly to mine. However, Formosa can produce about 2.5 million tons of coal annually, although local requirements limited the 1950 output to 1.4 million tons. Known petroleum resources are small, and significant discoveries in the future are unlikely. Domestic crude production in 1950, all from the Miao-li field north of Ch'ing-hsui, was only 22,700 bbl—only a fraction of normal domestic demands. This deficiency was accentuated by increased military needs. To supplement the supply in 1950, more than 400,000 bbl was imported, mostly from Arabia. The Kaohsiung refinery, with a daily capacity of 7000 bbl, treated all of the imported crude petroleum. Formosa also produced some natural gas, and it has a relatively large hydroelectric power potential amounting to 2 to 3 million kw. Only about 100,000 kw has been developed, mainly at Sun-Moon Lake (Jih-yueh-t'an) near the center of the island, but this is more than the present steam-generating capacity.

Primary iron resources are virtually nonexistent, except for mineral sands and unworkable pyrites. Before the end of World War II the iron and steel plants in Formosa depended upon iron ore from Hainan and the Chinese mainland, but now they must rely on iron materials from elsewhere as well as on the local supply of scrap iron to make the few thousand tons of finished products needed on the island each year. Coking-coal resources are also limited, totaling some 10 million tons. There is a small manganese operation at Mt. Hsimao near Su-ao, which may soon resume production. The Chin-kwa-shih (Kinkaseki) gold, copper, and silver mine near Taipei, perhaps has been the most important mineral enterprise in Formosa. It still has reserves of 5.8 million tons of 0.7 pct copper ore as well as 5 million tons of gold-silver ores. Before the war production of copper concentrates reached 6000 to 7000 tons annually, but the present rate is less than 500 tons. Major rehabilitation is necessary to raise the production to the wartime level, and the Chinese Government has recently asked ECA for assistance on the project. Gold production has likewise decreased; 18,232 troy ounces was produced in 1950.

An aluminum plant at Kaohsiung is equipped to produce 6000 tons of ingots, 4700 tons of sheets, and 1000 tons of aluminum ware, but the maximum postwar output was 2500 tons in 1948, when it operated on Japanese stocks and Malayan and Bintan ores. The plant virtually ceased operations late in 1950 because it lacked raw materials, and it probably will not resume operations until the newly discovered 200,000-ton bauxite deposit on Quemoy Island can start mining operations.

Formosa has 5 fertilizer plants at Keeling, Hsinchu, and elsewhere, capable of a combined annual production of nearly 50,000 tons each of calcium cyanide and superphosphate annually, but actual production during the last 3 years has been about 60 pct of capacity because of the unsteady supply of imported raw materials. The island is seriously deficient in fertilizer resources and cannot supply its fertilizer needs, which are about half a million tons a year. Large imports of fertilizers under the ECA have contributed greatly to Formosa's present healthy agricultural economy.

"Russia's Mineral Potential" Criticized

Russia's mineral potential is a secret that has been effectively kept by the Iron Curtain. There is no conclusive data and because of its extreme importance to the Free World, the subject is greatly discussed. Opinions range from those presented by Paul Tyler in his article "Russia's Mineral Potential," June issue of MINING ENGINEERING to those of Norman C. Stines, presented herewith. Mr. Stines is a consulting mining engineer of Los Gatos, Calif. He was in Russia, China, and Manchuria from 1909 to 1918 as a mining executive. From 1917 to 1919 he served as assistant military attache Intelligence Branch U. S. Army in Petrograd, Russia.

I HAVE become hardened to seeing all kinds of statements about Russia in the daily and weekly general press, but I did not think that our only mining organization periodical would be a party to spreading the myth of the "great productive ability" of Russia, by printing the article "Russia's Mineral Potential" in the June issue. It was hard to realize that I was reading a copy of MINING ENGINEERING. Only when I read of Tyler's long connection with the Bureau of Mines and in other federal government service does it become clear how anyone could have that view of Russia's mineral potential. Throughout the last twenty years I have become accustomed to the inaccurate reporting of Russia's mineral production in the publications of the federal government. ...

I feel that I know something of the mining industry of Russia, having been instrumental in finding and developing its greatest single copper orebody (I am not including any of the Soviet's claims of new discoveries)—The Degtiarsky mine in the Urals. I was not only managing director of the Sissert Mining District Co., which controlled that property, but I did the original geological examination and laid out and conducted the whole exploration and development program. I was also managing director of Altai Mines and when the revolution broke was exploring its Beleoufsky lead-zinc-copper-gold-silver property which has since become one of Russia's principal producers of lead and zinc. I was also managing director of the Lenskoi Gold Mining Co., a producer of placer gold in the Lena River area of Siberia. If my memory does not fail me its 1913 production of about 700,000 oz (not fine) of gold—worth at today's price approximately $22,000,000—is still unsurpassed by any single gold mining concern in any country.

This question of annual gold production of Russia has been a matter of controversy for years. Almost everyone but the United States' Government Bureaus, and its officials, recognizes that the Soviet's claims are untrue. ... in the middle thirties the president of the McIntyre Porcupine—J. P. Bickell — with a companion made a trip through Russia. The *Northern Miner* printed a communication from Bickell wherein he ridiculed the claims of gold production then made by the Russians. Engineers from other countries have also reported the same condition but not the Bureau of Mines or USGS.

As a matter of recorded production I am sure that a little delving into history will show that for many years previous to the revolutions of 1917 the annual rate of Russian production of gold was something less than 1,500,000 fine oz. Of course, those who do not know Russia will say that production could have been increased many times over that since under the Communists there has been an intense search for new deposits and this has been rewarded by numerous new finds. ... Such a statement might apply to almost any other mineral product, but not to gold.

In my residence in Russia from 1912 on, there was hardly a foreign expedition coming to Russia seeking gold deposits that did not first come to me for assistance of one form or another. ... Not one of them found anything worthwhile — certainly nothing that would bring the production to the 8,050,000 oz allowed by Tyler.

Our own companies ... also did their own searching and Lenskoi probably spent yearly in that search as much as all others combined. ... Our best find was the Kolyma Field ... (which) the Communists have boldly asserted ... was their find! Another promising field was the Aldan. But neither of these ... could bring the annual rate to any more than twice the pre-revolutionary norm. Personally I am certain that when the truth really gets out it will be found that Russian production has never exceeded 3 million oz a year and has averaged below that.

... I spent a long apprenticeship in searching for gold properties all over Siberia and the Urals and I am certain that there is no field in that area that has produced 200,000 oz or more that I have not visited. One of these searches took me into that country between the Amur and the Kamchatka Peninsula on those streams draining into the Sea of Okhotsk. I cannot claim that we visited every stream in that area but either I, or one of my assistants, visited every one over ten miles long. Did we find any gold? Yes, but never enough to warrant our return for more detail work. How could we have covered so much ground? Strange as it may seem we never could arrive on any stream where we would not find the dumps of shafts sunk by predecessors—the trees growing on such dumps were from 15 to 35 years old. The excellent Chinese and Korean (never Russian) prospectors had by those or more years beaten us to it.

Their work made it very easy for us. All we had to do was to find bedrock on the dumps to know that the pioneers had reached the bottom of the gravels. The washing of all or portions of those dumps would reveal the average gold content of the material they had passed through. Most of those gravels were barren. We soon learned what types of bedrock would mean gold in the gravels or on the bedrock, and that also made our work less difficult. (... the gravels ... are permanently frozen, so prospectors would have no difficulty with water.) Decidedly no, the Russians have not found deposits that would allow them to expand their annual production from 1.5 to over 8 million oz.

... Around Orsk were some productive lode mines and the Urals yielded a few. In all of Siberia there was only one lode gold mine that I knew of that could be said to be a large one—that was exhausted years ago. Even in that very prolific producer of placer gold, the Lena region, none was found—like the Yukon and the Fairbanks areas of Alaska.

... During the '30's, I happened to be doing some research work in a New York laboratory. One of the proprietors, knowing that I had lived long in the former Russian Empire, one day told me that that afternoon a Russian was coming to the lab. He was one of the top men in the Government bureau that was directing whatever gold dredging (was) then being done. ... The proprietor then asked me to see him and attempt to gain certain facts the proprietor sought. This, if it was distinctly understood that the Russian was not to know that I had been in Russia or could speak Russian, I agreed to do. ... What the proprietor wanted cleared up was soon accomplished. Then the Russian was told that I had done considerable dredging work in the United States and would probably be interested in hearing about that work in Russia. ...

The Russian began by saying that before Communism had come to Russia, there were no electrically driven dredges there. Note—there were two at Orsk in the Far East, one on the Sissert, and two or three on the Nikolai-Pavda Estate in the Urals, to mention only those that now come to my mind.... the Communists had put in several....

"There are the dredges on the Miass Estate in the Urals," he stated. He added that they were working very rich ground. I had prospected the streams on that property and knew that the frozen gravels did not contain over 2 or 3c worth of gold per cu yd. The ground would have to be thawed before it could be dredged, with a total cost of something over 20c per cu yd for the thawing and a further 10c for the actual dredging. ... I realize that this excess cost did not bother the Russians as what they urgently needed—and still do—was foreign exchange and at no matter what cost. But the annual rate of gold production cannot be increased very fast on 2 or 3c worth of gold from each cubic yard. Even one of Yuba's largest dredges handling 5,000,000 cu yd annually could not produce over $150,000 worth!

"Then in the Semipalatinsk area are also great rich fields," he added. There I was again at home. I had spent several months in that area and could not find anything better than 6 or 7c per cu yd. Of course the Russian could dredge that at less loss, but could not thereby greatly increase the annual rate of production.

To be sure that there could be no misunderstanding about the entire lack of electricity driven dredges in Russia before the Communists put them there, I returned to that phase. I wanted to learn if the man were lying or had been fooled by the Communists' own propaganda. When he emphatically repeated that statement I forgot that I was not supposed to know Russian and had never been there, and rather angrily replied in Russian: "That is not true!" As the cat was out of the bag, we then conversed in Russian.

Out of that discussion it developed that he had been taught that in pre-revolution days there were no such organizations in Russia as a Russian Institute of Mining and Metallurgical Engineers; Civil Engineers, Mechanical Engineers, etc. I told him that there were and that they had issued their publications...

Now about the copper production. I very much doubt that it is 275,000 of our short tons per year. I do not doubt that competent exploration engineers and geologists could find the deposits but I am certain that the Russians could never develop them and bring them into a production rate that would achieve those 275,000 tons per year. Let me give you another real experience of which I know....

It will be recalled that during the '30's the Russian Soviet authorities sought and took into Russia many engineers and metallurgists to rehabilitate their metallic—particularly nonferrous—production. A party comprising a chief and several assistants reached Moscow. There as an interpreter the Soviet authorities provided a female mining engineer. The party set out for the Urals and visited the Verk-Isetsky, the Degtiarsky, and the Kalata installations.

The latter is the oldest of the plants to be installed by English-speaking engineers and metallurgists, and under them it had the greatest annual rate of production. But the Russians had been unable to get it again into production.

Before the American party could make an intelligent start, the plans and sections of the underground workings would have to be available. Resident on the property and at Kalata itself was a party of Russian engineers and metallurgists sent there to resume the production of copper. Appeals were made to them to produce a set of the maps. These Russians explained that they had none and added that when the former American staff, after the revolution, left Russia, its members either took with them the tracings or destroyed them and all prints....

The interpreter did not believe that story and suggested that the former Russian general manager and metallurgist be located in the prison camps to which she was certain they had been sent. These men were found. In time, two very wretched and completely broken in health and mind individuals were brought to the property. The former general manager (Ivanoff whom I very well knew) was too far gone to be of any assistance. The metallurgist (Kolassmikoff, whom I remembered as about the only Russian who really knew the metallurgy of copper, learned from his apprenticeship under his American tutors Walter Perkins and T. Jones) was in fair shape and knew the answers. He told the Americans that he was the last of the former staff to be arrested and, when he was taken, all the drawings—tracings and prints—of both the mines and the plant were in the engineering office "over there", pointing to a log building apart from the others.

This started the interpreter thinking. She found the old man who acted as watchman and in some way persuaded him to allow her, in the evenings, to get inside. She would enter, take out enough of the drawings for that night's work, bring them to the office, where the Americans would trace them. In time they obtained all they needed and proceeded to reopen the properties and put the smelter in condition for operation. Seeing this being done, the Russian engineers, who had been living there and had reported that the mines could not be reopened, began to stir and finally found how the plans had been obtained. They then reported the theft by the Russian traitor and demanded that the local police arrest her and have her shot for treason. One of the workmen, all of whom were for the Americans, for they remembered how conditions were when the Americans before the revolution ran the plant, told the interpreter of what was doing and advised her to flee....

What I really want to show by this episode is that the Russian technical men could not even reopen their own developed properties and when Americans were succeeding, they were vindictive and jealous enough to accuse the Russian girl, who had assisted the Americans, by declaring her a "traitor." How can it be traitorous for a Russian to want to help someone produce copper for the Soviet? That only shows once more the cockeyed thinking of all those close to the Communists.

I note one more statement that shows Tyler has no conception of the situation in Russia—either pre or post-revolution. "Enough is known of the geology of this vast area to enable some evaluation of official claims of exploration achievements." I can best make this clear by picturing how a Russian geologist studies geology.

In an actual case in pre-revolution days, a geological party appeared on one of the estates in the Urals. Headquarters were set up in one of the most comfortable homes at the headquarters. A map of the estate was spread on one wall and certain spots were marked. The local foresters (for each of the estates in the Urals had a complete forestry staff to govern cutting of the timber so the regrowth would equal the annual cut, this system having been put in force by Germans in the 17th century) were called in and each of them told to proceed to a certain designated spot and there get a rock specimen and bring it in. Any one knowing the Urals realizes that outcrops are few and far between. The country is well rounded—very old topography. And any one who knows Russians knows no Russian, even a forester, would return empty handed. If there were no rock at the designated place, he would get one as near as he could—what difference if it were one or two thousand feet distant?—and bring that in, neglecting to tell the chief what had been done. How good is a geological study made in that way. How can we know enough "of the geology of this area" when any description of it is based on a survey made in that manner?

Please bear with me just one paragraph more. Note that production of molybdenite increased 60 times; copper, three; tungsten, nine; and tin, 18. Do such increases sound reasonable in a country where labor efficiency is only about 10 pct of that in other parts of Europe. It certainly does not. It just can't be.

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CASTLE DOME Overcomes Increased Truck Haulage Grades

by J. C. VAN DE WATER

THE original trucks at Castle Dome were 30-ton Knuckeys with 150 hp, H-series Cummins engines and 15-ton Euclids with the same engines. The Euclids were used early in the operation, working with 2½-yd Diesel shovels on stripping and later in opening new benches where the lack of room favored the smaller trucks because of their shorter turning radius. They have also been used on road building and similar jobs. On regular production, where the 30-ton trucks can be used, they are not economical.

The 30-ton Knuckeys hauled the bulk of the stripping and all of the ore through 1948. During this time grades were level or favorable to the loads except for a short haul of +3 pct in the latter part of 1948. On the +3 pct haul about 5 mph was top speed and any steeper grades were out of the question. With the prospect of longer 3 pct hauls and some long 8 pct hauls in the future, 5 Kenworth trucks were purchased late in 1948. The Kenworths are basically the same design as the older Knuckeys but with a 275 hp Cummins NHBS engine and correspondingly heavier clutch, transmission, and differential. These trucks performed so well that 5 of the Knuckeys were rebuilt with the 275 hp engines, and the same clutches, transmissions, and differentials, as the Kenworths. These proved equally satisfactory and these 10 trucks now compose all of the active haulage equipment.

Although the Kenworth and Knuckey trucks don't look exactly alike, they are practically identical mechanically so that the following description covers both trucks.

Power—Cummins NHBS 275 hp supercharged engine.

Clutch—Spicer 14-in. two plate.

Main Transmission—Spicer, 8241.

Auxiliary Transmission—Spicer Brown-Lipe, 8031 C

Differential—Timken double-reduction FU 79

Drive—Chain, from jack shafts on differential to each rear axle, drive sprocket 19 teeth, driven sprocket 44.

Front Axle—Timken tubular 25,000 lb

Steering Booster—Vickers.

Frame—Flexible type made of three 11x3½x⅜-in. nested channels.

Body—Box type 22-yd or 30-ton capacity equipped with 15 manganese plate liners bolted to bed.

MR. VAN DE WATER is Mine Superintendent, Castle Dome Copper Co., Inc., Miami, Ariz., and is an AIME member.

Hoists—Heil telescopic, 4 stage on Knuckey and Atlas telescopic, 4 stage on Kenworths.

Brakes—Air, Westinghouse Bendix.

Tires—10, 1400x24, 20 ply.

The separate parts as noted above with the maintenance problems of each are as follows.

Power—When the 275 hp engine was first used, excessive piston breakage, trouble with wipers coming out and valve seats burning occurred. Piston and wiper trouble was overcome by using a heavier wrist pin and a new type piston without a wiper. To eliminate the valve trouble, the regular valve seats were removed and replaced with stellite inserts.

Clutch—Normal maintenance.

Main and auxiliary transmission—Over the past year, the main transmission averaged one overhaul for every 12,000 miles and one on the auxiliary for every 14,000 miles. These jobs ranged from replacing oil seals to replacement of the assembly.

Differentials—Replacement of ring and pinion gears averaged one in every 37,000 miles up to the time that a long +8 pct haul was started. On this haul the replacement rate jumped up to one in every 7000 miles. Overheating caused by the heavy grade was suspected. Temperatures were checked but none were found to be over 200°F which is normal for the unit. Attempts are being made now to decrease the failure rate by lightening loads, eliminating all unnecessary shifting, and by closer attention to adjustment and installation in the shop.

Drive—Drive sprockets and chains run about 7000 miles per replacement; driven sprockets about 18,000 miles.

Front axle—No maintenance.

Steering boosters—Only minor adjustments and repairs.

Frame—This flexible type of frame begins showing fatigue cracks between the hoist saddle and jack shaft mounting after 12 to 18 months' service. These are welded and patched as they occur. Frame rails probably will have to be replaced at the end of about 3 years' service.

Body—Replace 3 back liners every 18 to 24 months.

Hoist—Since installation of the plate bed liners, the beds only have to be raised through the third scope to clear the load and hoist trouble is practically eliminated.

Brakes—Full braking on all wheels, controlled by foot and hand valves, plus emergency tanks for each rear axle. Recently an air emergency brake on the

drive line located just in front of the differential housing was installed. No trouble except occasional air leaks has occurred.

Tires—Data on 72 tires of various makes is as follows:

Number Tires	Original Miles	Average	Number Recaps	Recap Miles	Average per Recap	Total Miles	Total Average
72	906,344	12,588	63	464,363	7,371	1,370,707	19,038

Lubrication—Complete lubrication every 48 hr. Engine oil change every 150 hr. Hoist, fuel, and chain oilers filled each shift. On day shift, each truck is brought to grease pit and while it is being serviced, a mechanic checks for loose bolts, etc., and does any minor repair work necessary. This preventative maintenance has eliminated a large number of shop repair jobs.

During the past three months 933,113 tons of ore have been hauled an average distance of .975 miles with an average vertical lift of 162.9 ft. Road grades were level, +3 pct and +8 pct. All haul roads included from 600 to 3000 ft of +8 pct grade. Haulage costs during this period were divided as follows:

	Percent
Operating labor	20.8
Operating supplies: tires	12.9
petroleum products	8.8
miscellaneous	2.6
Total operating	45.1
Maintenance: labor	21.1
parts	32.4
miscellaneous	1.4
Total maintenance	54.9

OPERATING EFFICIENCIES

Trips per shift	31.7
Tons per trip	32.9
Tons per shift	1045
Average haul-mile	1.8 (Rd. Trip)
Ton-miles per shift	941.3

Books for Engineers

Soil Testing for Engineers. By T. W. Lambe. Published by John Wiley & Sons. 165 P. $5.00.—Filling a need for a text for the teaching of soil testing in the laboratory, this book is also of value as a reference for practicing engineers and for personnel in soil laboratories. Following an introductory chapter on general laboratory procedures are chapters devoted to the individual laboratory soil tests which are commonly employed. Apparatus, supplies, recommended procedures, discussion of procedure, calculations results, and numerical examples are provided for each test. Brief derivations of formulas and explanations and discussions of calibration procedures and special techniques are given in the appendix.

Mesabi Pioneer. Copyrighted 1951 by the Minnesota Historical Society.—Publication of this book has been made possible by a generous gift to Society from the E. J. Longyear Co. of Minneapolis. This book covers the evolution of the Mesabi Range from a wilderness into the nation's greatest iron producing area. In this book is found the first hand experiences of Edmund J. Longyear who saw the development of the range from its beginnings, and who himself took a notable part in it. In 1890 Edmund J. Longyear sank the first diamond drill hole on the Mesabi. During the next two decades he sank pits and holes—7133 of them—throughout the length and breadth of the Mesabi Range, exploring for iron ore. He saw roads and railroads built. He watched the range communities—Hibbing, Chisholm, Biwabik, Mountain Iron and the others change from exploration camps to drill camps, then to mining camps, and finally to villages and towns.

Structural Geology of North America. By A. J. Eardley. Published by Harper & Brothers, New York, 1951. 624 P. $12.50.—This book is a comprehensive and detailed account of the structural evolution of the North American continent in post-Proterozoic times. It considers the formation and constitution of the mountain systems, basins, arches, and volcanic archipelagos; the beveling of the highlands and the filling of the basins. It not only chronicles the crustal unrest but summarizes the supporting evidence. Numerous illustrations and a 700-item bibliography are included.

Mathematical Solution of Engineering Problems. By J. Jennings. Published by E. & F. N. Spon Ltd. of London, 1951. 208 P. 25s.—This book has been particularly designed and written for the use of technicians who are familiar with basic mathematics but require guidance in effective practical application. Special features are the chapters on the construction of the nomogram, on statistical methods, and on dimensional analysis. The importance and utility of approximate methods of solution have been emphasized and practical illustrations given.

Coal Mining. By I. C. F. Statham. Published by English Universities Press, 1951. 564 P. 22s. 6d.—Written in non-technical language, this book describes the past and present of the coal-mining industry of Britain. It covers the surface plant, shaft and shaft-bottom arrangements, working the coal, mine structures, underground transport, ventilation, lighting, drainage, power, safety, health and training in this industry. Numerous photographs and drawings illustrate the text.

With Rod and Transit. By James B. McNair. Published by author, 1951. 267 P.—This book covers the engineering career of Thomas S. McNair. His education, teaching career, family background, engineering and civic background are all discussed. Thomas S. McNair was one of the organizers of the American Institute of Mining and Metallurgical Engineers. Under his supervision was constructed one of the longest (at that time) tunnels in the world, which was also very accurate as to line and grade. To him are due the invention of a new mining transit and many improvements in coal mining such as the use of the measurement of vertical distances by altitude. His career began with teaching, then railroad surveying, then canal engineering to be followed by railroad and coal mine engineering. His greatest contributions were made in mining engineering, in which occupation he spent 40 years. He came upon the field at a period when specialties were unthought of, when circumstances made him at one time a hydraulic engineer, at another a mining engineer, opening the coal fields of Pennsylvania, and again reverting to canals, to railroads and to waterworks. He was one of those whose names are inseparably connected with the development of the resources of this country, and with the rapid progress made in means of transportation.

The Metal Mining Industry in Japan

by Robert Y. Grant

This study outlines the history of metal mining in Japan and the characteristics of the industry as they existed from 1925 to 1945. Mining and milling operations are described together with the role of the Japanese Government. A description of post-hostilities conditions and the present state of the industry is included.

MANY of Japan's larger and better known mines date back to the early centuries of the Christian Era. The first stimulus to mining in Japan came through the growth of Buddhism, A. D. 600 to 700, and the attendant need for copper and other metals for temple ornaments and statues. During the Feudal period, 1333 to 1568, mine operations were expanded to meet the demand for ornament metal and for financing of campaigns. After removal of the seat of government from Kyoto to Tokyo, about 1600, the Tokugawa Shogunate placed great emphasis on mining, and many new mines were opened, some of which are still operating.

Toward the end of the Tokugawa period, mining had declined sharply from its earlier peak. The easily worked, high grade portions of the orebodies had been exhausted, and the mining techniques then available could not exploit the deeper, lower grade ores. Similarly, extraction of metal from the sulphide ores presented a major problem.

With the Meiji Restoration in 1868 and the rapid opening of the country to western ideas came a rejuvenation of the mining industry. Mining, milling, and smelting processes were greatly improved, resulting in the extension of development work into hitherto inaccessible portions of orebodies.

The demand for metal during World War I stimulated activity throughout the industry; copper mining, particularly, expanded to a point where Japan ranked second among world producers.* During the depression following the war, mining underwent a severe contraction in activity, recovering slowly during the late 1920's only to undergo a further setback in the worldwide depression which started in 1929. Mineral output declined in 1930, but the beginning of Japanese expansion, 1931 to 1932, brought about an increase in the output of mine products starting between 1932 to 1934. Production continued to rise, the peak output being attained 1939 to 1943 in virtually all commodities. The greatest production of iron ore, chromite, and manganese was not experienced until 1944 when blocked imports, coupled with strong demands for steel, provided the needed incentive.

Through necessity, Japan produced much of her mineral needs from her own mines at high cost during World War II. Taking the 1932 to 1936 period as an average, however, Japanese mines produced only 16 pct of the iron available to industry, 8 pct of the lead, 33 pct of the zinc, and 68 pct of the copper.

Nearly all of Japan's output of gold, silver, copper, lead, and zinc between 1925 and 1945 came from 30 to 49 pct of the mines for which production data are available; for example, 66 of 174 gold-silver mines accounted for 98 pct of the output of this group. Further analysis of mine output indicates that a large part of Japan's metal output was derived from a few large mines; about 10 gold-silver, 15 copper, 3 lead-zinc, 5 manganese, and 2 iron mines account for most of the production. Many of these mines have been operated for 200 to 400 years. With the

* John J. Collins: *Copper in Japan*, Natural Resources Section Report No. 106.

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exception of the mines in Hokkaido, where settlement took place late, four of the five largest gold-silver mines were opened before 1660 and seven of the eight largest nonferrous metal mines were opened before 1690.

With increasing supply and manpower difficulties, gold mining was drastically curtailed in 1943, largely by cutting off allocation of supplies. About 50 pct of Japan's gold mines were closed and others suspended milling. Many gold mines were stripped of equipment, which was shipped to other mines in Japan or Korea.

Postwar recovery of the mining industry has been impressive, and much of it has been due to substantial loans from the Japanese Government and from funds supplied through various types of subsidy. Withdrawal of government loan support in the latter part of 1949 was more than compensated for by rising metal prices starting in mid-1950.

Between 1925 and 1945 Japanese metal mining operated with an abundant supply of inexpensive labor, a ready market for minerals, government controls, and a fair margin of profit guaranteed by subsidies. With conditions such as these, a study of the characteristics of the Japanese mines and the mining industry during the pre-1945 period will help toward an understanding of the industry in the post-Surrender days. Most of the operational methods discussed below are still in use.

Mining

Ore Grades: During 1925 to 1930, most mines operated on relatively high grade ore, but the better deposits were quickly exhausted, and from an examination of production records it is apparent that most mines in Japan handled low grade ore during the late 1930's and during the period of hostilities. As an example, of total reserves of 25,500,000 metric tons reported by the 38 largest gold-silver mines in 1945, 15,000,000 metric tons averaged 3.0 to 3.9 g of gold per ton, and 5,000,000 metric tons, from 2.0 to 2.9 g per ton. In one of the two largest copper mines, the grade of the milling ore dropped from 0.79 pct copper in 1940 to 0.68 pct in 1945. In the other mine, the crude ore grade dropped from 1.69 pct copper in 1940 to 0.91 pct copper in 1945. These copper mines are underground operations, not open pit.

Except in rare instances, geologic mapping and study in the mines were wholly inadequate to meet the need for more ore. Detailed geologic maps were virtually unknown, and studies of alteration or structure as guides for finding ore were nonexistent.

Extraction, Timbering and Drainage: In spite of low grade ore, mining practice in the metal mining industry was standard except for employment of a large proportion of hand labor. Drills were, in most cases, copies of American equipment, but lighter in weight. The use of wet stopers was almost unknown. A small amount of experimental work on detachable bits was done during the early 1940's, but since bits were not perfected, mines used standard drill steel. Cap and fuse blasting was the universal practice. Because of the lack of control in drilling and the mining of narrow veins, considerable overbreaking was experienced in almost all properties. Smaller producers or operators of mines working irregular or small deposits employed hand methods almost exclusively.

Almost no mechanical loaders were used, not even in tunnels or development headings; hand-loading was the rule, with hoe and pan arrangement instead of the conventional shovel or scoop. The loaders rake the muck into a small two-handled pan with the hoe, lift the loaded pan, and empty it into a car. Timbering followed western practice, but almost all timber framing was done underground, with pine preferred.

Pumping practice was normal. Some Japanese mines are extremely wet, and in a few instances hot springs complicate the drainage problem, but the sharp relief facilitates mine drainage.

Haulage and Hoisting: Hand-tramming was practiced to a great extent, being used at mines where maximum annual output did not exceed 200,000 metric tons. Battery locomotives often were used for main haulage in the medium-sized Japanese mines, but in the few mines of over 300,000 tons per year capacity, trolley locomotives were the rule for main haulage; battery locomotives were employed in collecting. In most instances, the cars, including those used on main haulage, had a capacity of from 1 to 2 tons. In some larger mines 3-ton cars were used.

The largest amount of crude ore handled anywhere was in an iron mine which produced 1,147,000 metric tons in 1944. Most mines produced less than 500,000 tons of crude ore. The maximum output of the ten major producers is given in Table I.

Although in a few mines, skips were provided for main hoisting, most mines hoisted cars.

Mine Safety: Safety practice in the metal mines was not satisfactory by western standards, due as much to indifference on the part of the individual employee as to failure of management to require safe operating practices. The lack of interest in safety is reflected in the lack of information on pre-1945 accident rates available from Japanese Government records.

Mining Costs: Cost data are unsatisfactory because of poor cost accounting methods and because of incomplete information on labor force and on subsidy support. The operations of a few of the base metal mines showed an average mining and exploration cost amounting to 63 pct of the total, with hoisting and haulage accounting for 24 pct; pumping, 4 pct; and maintenance and timbering, 9 pct. Between 1930 and 1945 labor in mining operations accounted for 50 pct of the total cost, minerals about 25 pct, and electric power about 5 pct. The remaining 20 pct was divided among overhead charges and various incidental expenses. Among gold-silver mines the same cost breakdown was the rule.

Ore Dressing

Flotation plants were established at most of the larger nonferrous metal mines, but some produced direct smelting ore shipped after sizing and picking. Almost every mill contained a large hand-picking section. Cyanidation was employed at the gold-silver mines.

The largest flotation plant among the base metal mines had a capacity of 3000 tons per day. Among the cyanide plants the largest mill had a capacity of 2000 tons per day, in a mine which produced 740,211 tons in 1942, the year of largest output. Many of the smaller gold-silver mines employed hand-picking as the only concentrating process. Mill equipment was of American design or, in some cases, of American manufacture.

Efficiency: Although of doubtful accuracy because of poor sampling practice and the lack of appreciation by management of importance of maintaining

mill records, the reported average recovery (1930 to 1945) in cyanide plants of 90.36 pct for gold and 75.-58 pct for silver is of interest. Metal recoveries in flotation plants were variable, depending upon the type of ore being processed. Copper recovery (1930 to 1945) is reported to have been as high as 92 pct. A low of 65 pct recovery was reported for a complex copper ore containing a large amount of clay. Lead recovery varied between 65 and 90 pct. During World War II, the scarcity of reagents and overloading of mills resulted in a lowering of recovery.

Costs: Costs in Japanese mills did not vary widely from those experienced by American mill operators, with a crushing and grinding cost of about 50 pct of the total, and flotation and hand-picking, 30 to 40 pct. Japanese tailings disposal expenses were high, about 6 to 7 pct of the total. Between 1930 and 1945, labor in Japanese flotation plants averaged about 15 pct of overall costs. Materials cost operators 50 pct, and power 18 to 20 pct of the total. A slightly different pattern was found in the cyanide plants where labor costs amounted to 25 pct, materials about 33 pct, and power about 15 pct of operating costs.

The sale of ores and concentrates to smelters was governed by the same general conditions that existed in the United States and elsewhere. Slightly less attention was given to penalties or other charges than has been the practice in the United States smelters, but this is in accord with the general lack of close control. Also, minor items of cost were of less concern to management.

Labor

Causes of differences between American and Japanese mining are to be found in the social organization, the educational system, and economic conditions that exist in Japan. The Japanese metal mine worker usually is a native of the area in which the mine is located. If a mine is abandoned, the workers remain in the area, turning to full-time farming for support. In the older mines, as many as three or four generations of the same family may have been employed in the mine.

An unsatisfactory ratio between the number of underground and surface workers in Japanese mines always has been reflected in excessive office staffs. By 1945 only from 30 to 40 pct of the total labor force was employed underground. This was not caused entirely by a shift from underground to surface but in part by an increase in one segment of the labor group. A sizable increase in service personnel was required with the introduction of food rationing and the expansion of welfare activities occasioned by the departure of heads of families or wage earners for the military services.

Table I. Maximum Output of Major Producing Mines in Japan, 1925 to 1945

Type of Mine	Year	Maximum Crude Ore Output, Metric Tons
Iron ore	1944	1,147,000
Copper	1942	1,007,000
Lead-zinc[b]	1943	983,000
Copper	1944	893,000
Copper	1942	839,000
Gold	1942	740,000
Copper	1943	696,000
Gold-copper	1942	656,000
Pyrite	1938	600,000
Gold-copper	1942	506,000

[a] Source: company records.
[b] Two adjacent mines under same management.

Most Japanese mines employ many women workers, particularly in surface installations. Women commonly were employed underground for haulage and similar tasks in the 1925 to 1945 period and particularly during World War II. Korean and Chinese laborers began to be employed near the end of the 1930's. By 1944, in the Hitachi copper mine, one of the largest in Japan, 1089 workers of a total labor force of 3154 were Koreans. Prisoners of war were used in many of the larger mines during the latter part of the war.

The wage structure followed the usual Japanese wage pattern. The worker received a basic wage plus a number of bonuses and allowances. Attendance allowance, efficiency allowance, high price allowance, family allowance and underground work allowance are a few. Special allowances were expected for marriage, the birth of a child, and deaths in the family, and at the end of the year. In addition to the wages, the Japanese mine worker received benefits from the company in the form of low prices for many items of food and clothing.

Between 1925 and 1945 surface workers received only 60 to 80 pct of the wage of the underground workers. Women workers in four representative Japanese mines received an average wage only 37 pct of the wage received by the male workers who, in many cases, were doing the same or lighter work.

Unions were organized, but rigidly controlled by the companies; they were largely organizations permitting a more convenient hold over the mine population.

On the other hand the paternalistic pattern of all Japanese industry was found in mining as well. The operator was expected to assume responsibility for support of the families in case of injury to workers. Formal agreements were not normally established, nor were compensation payments standardized although compensation laws existed. Pay deductions for welfare or compensation were rarely made. Under such a system, as mentioned above, compensation to an injured worker often took the form of continued support to the family, or in a case of permanently incapacitating injuries, light work in the office was often provided. When mine operations were suspended for a period, the operators continued to support most of the workers and families.

Government and Mining

Prior to the mid 30's the Japanese mining industry was relatively free from government interference, but as requirements for minerals increased, controls began to be introduced. By 1938, most of the mineral and metal prices, as well as the supply of metals, were controlled. Every phase of the industry was supervised by the Ministry of Commerce and Industry† through the Mining Bureau. Authority for such supervision was present in the Mineral Industry Law, the basic mining law of Japan, as well as in emergency legislation. A number of semigovernmental organizations established during and after 1938 permitted the government even greater control by allowing it to enter directly into operations. It was through these groups also that much financial support was provided to the mining industry.

Control Organizations: The most important of the

† Now Ministry of International Trade and Industry.

semigovernmental organizations in relation to the nonferrous mining industry was the Imperial Mining Development Co. Ltd., organized in April 1939. The Japanese Government held 50 pct of the stock of the company. Imperial was authorized to enter into any activity related to mining, including exploration, mine operation, granting of loans, and even operation of such auxiliary facilities as aerial tramway manufacturing concerns. The Japanese Government was to provide funds where necessary to guarantee a 4 pct profit to shareholders during the first 5 years of operations and 6 pct thereafter.

A second organization that worked closely with the Mining Bureau was the Metal Mining Control Assn., established in 1941, with a membership made up of nonferrous mining companies. It was formed to work out the solution of production problems and was an outgrowth of four control associations, the Copper, Lead-Zinc, Tin, and Mercury Control Associations which had functioned since March 1938. A Metal Distribution Co. which operated in conjunction with the Metal Mine Control Assn., was the distributing agency for nonferrous metals.

Government control in the ferrous field was provided by the Iron and Steel Assn., and by such groups as the Imperial Manganese Co. (later the Imperial Manganese and Chromite Co.) with functions similar to those of the Imperial Mining Development Co. and the Metal Mining Control Assn. Tungsten, molybdenum, pyrite, and gypsum were brought under control in 1943 when the Imperial Manganese and Chromite Co. was redesignated the Ore Distribution Control Co.

Financial Support: An intensive program of monetary support was started in 1938, partly to stimulate production and partly in apparent recognition of the lower grade ore, smaller reserves, and rising costs. The subsidy program also permitted the government to maintain control over domestic price levels by covering operating losses through subsidies rather than through price raises. The effect of the strict control over prices of metals, in spite of increasing costs, is illustrated by small rise in the official copper price between 1938, when it was Y1,050 per ton, and 1945 when it had risen to Y2,060 a ton. This is in marked contrast to the situation after 1945 when the price of copper rose from the 1945 price of Y2,060 to Y181,060 a ton in 1948.

The subsidy program gave direct financial support to the mines covering up to 50 pct of exploration and mill construction costs. In addition, subsidies were provided on gold and other metals. With the yen price of gold at Y3.85 per g, production bonuses from Y0.80 to Y3.30 per g were paid. Similar bonus plans were started in 1943 covering other metals. The amount of money paid to a mine had little or no relation to metal output. In 1943 one marginal property received Y1,279,809 in metal price bonus funds for a production of 624 metric tons of copper in concentrates; during the same year, a second mine received Y1,327,788 while producing 8343 tons.

Funds also were provided to the industry in the form of loans to the mining companies from the semigovernmental organizations, such as the Imperial Mining Development Co. Between 1939 and 1945, during a time that the yen-dollar exchange was about $0.25 to Y1.00, about Y333,000,000 in loans were made to the mining industry; most of this has not been repaid. Large amounts of money were made available to metal mining through the purchase by the Imperial Mining Development Co. of gold properties after suspension of gold mining in 1943. More than Y230,000,000 was paid for the mines although only Y4,000,000 was made available as cash. The remaining funds were deposited in blocked accounts. The blocked funds could be utilized by the mine owners for the payment of taxes and other government obligations or for the support of the operation of lead-zinc or copper mines. This effectively drained the blocked accounts. Many of the mining rights so acquired are still held by the government.

Table II. Mine Production by Company, In Pct of Total Output 1948 Calendar Year[a]

Firm	Copper	Lead	Zinc	Pyrite
Mitsui Mining Co.		41	64	
Mitsubishi Mining Co.	28	36	20	5
Nippon Mining Co.	27			10
Seika Mining Co.	8			5
Nippon Zinc Mining Co.		5	5	
Furukawa Mining Co.	13			3
Dowa Mining Co.	8			34
Matsuo Mining Co.				29
Other	16	18	11	14
Total production,[b] metric tons	26,000	6880	33,640	1,138,100

[a] Source: company records.
[b] Metal content of concentrates except for pyrite which is expressed as concentrate.

The total amount of financial assistance made available to the nonferrous mining industry through 1939 to 1945 cannot be determined accurately, because of loss of records and confusion in respect to financial transactions. Available data on a sizeable segment of the industry indicate that at least Y800,000,000 were introduced as new capital, and it may be that half again as much was furnished to the industry. The importance of such financing cannot be discounted. Records of one copper mine show that 50 pct of the gross income of the mine was derived from the Government in 1943.

During 1945, mine output dropped sharply, with almost complete suspension of operations after the end of the war. Most of the gold mines failed to reopen, with only 14 of the 40 largest being in operation in 1946 and only 18 in 1947, falling off to 16 in 1948. Nonferrous metal mines, for the most part, resumed operations, with 26 of the 32 largest in operation in 1946; 27 in 1947, and 26 in 1948. Small mines began operations also, but the number was greatly reduced; by December 1948, only 159 copper mines reported production, and 177 in 1949 in contrast with over 300 during 1935 to 1940. Very few of Japan's iron mines were able to reopen because of the low grade of ore reserves.

In spite of slow initial recovery caused by financial difficulties, labor problems, shortages of supplies, and uncertainty within management circles brought about by the threat of deconcentration, by the end of 1949 substantial recovery had been made. Production of most metal was rising and plant expansion was being undertaken. Although some setback was experienced with the removal of price controls and elimination of government loans, rising metal prices and increasing demand for metals of all kinds beginning in the spring of 1950 gave added impetus to the recovery movement.

At present most metal mining companies show a clear profit from their operations, and mines which have not operated since 1944 and 1945 are being reopened. Competition for ore supplies is growing,

with some companies constructing new mills or adding to old ones for the primary purpose of treating custom ore.

The worldwide scarcity of metals and strong competition in world markets for ores and concentrates make it probable that Japan's industrial machine will find it necessary to depend in large part upon the domestic mining industry for basic raw material supplies. For this reason, in spite of low grade ore and small deposits, Japan's metal mining industry should enjoy profitable conditions as long as rearmament programs continue to occupy their present important position.

Equipment and Supplies

With a fuel shortage during 1946 and 1947 curtailing industrial recovery, machinery and equipment were difficult to get and expensive. Fuel, lubricants, steel pipe, air and water hose, electric wire, steel plate, and conveyor belts were in short supply.

The 16 pct of all mine equipment deadlined on Jan. 31, 1947, for lack of spare parts and material, was one effect of the shortage.‡ Other equipment was being run at reduced levels to avoid the frequent breakdowns resulting from capacity usage. Operators reported that at least 70 pct of all equipment was at least 5 years old. Coupled with the shortages was the poor quality of much of the material available; as an example, wire rope was being worn out in from 6 to 9 months.

The shortage of supplies continued through 1947. The nonferrous metallurgical industry reported that the items found to be short in the mining industry in 1946 were still out of stock at the end of 1947.§

By 1949 coal production had increased to a point where most industries were receiving reasonably adequate supplies. Equipment and materials became available in ever increasing quantities with the result that long delayed rehabilitation and development work could be undertaken. In addition to the normal replacement of worn machinery, some properties undertook large scale changeovers in mining methods. The use of scrapers was started at several mines; new drilling methods were instituted as new and better drills became available. A completely new underground ore-handling system and crushing plant were installed at the Yanahara pyrite mine, Okayama Prefecture, the second largest in the country. New mills were built and the flowsheets of old mills revised to include heavy media or other newly developed equipment. Although by July 1, 1951, some effect of the Korean affair was noticeable in the availability of equipment, in large measure little difficulty was being experienced by mine operators in purchasing new machinery.

Unions and Labor Supply

Coupled with high prices, shortages, and high costs, labor difficulties contributed to the slow recovery of the mining industry. Metal mine unions have not been as troublesome as coal mine unions, but as might be expected, with the introduction of the new concept of labor unions free from control by management, confusion arose with respect to the rights of management and unions. At first union officials received full-time pay from the mining companies, although they spent all their time on union activities. Management personnel were forced to spend many hours, which could have been spent to greater advantage in supervision operations, negotiating over union complaints, most of which were minor and many of which had to do with activities properly the concern of management. Labor supply is adequate for current needs.

Unions: Most of the workers and unions belong to the All Japan Federation of Metal Mine Workers, organized in May 1947 with 126 member unions and 70,000 members. It had a claimed membership of 83,000 in December 1948. Leftwing influence has been strong but not dominant. Following early efforts to achieve industry-wide action, which failed to materialize, on wages and working conditions, recent union bargaining has been limited to discussions and several work stoppages which rarely spread beyond unions within a single company. Strikes affecting the entire industry have been held, but these also were merely more or less simultaneous action by single mines, in some cases apparently for the sole purpose of saving face by not being left out of the strike action.

Labor Supply and Distribution: At the end of 1950 the Japanese Mining Bureau reported that 86,633 workers of all classes were employed by the metal mining, smelting and refining industry, not including the iron and steel plants. In 1949, 80,498 were employed; of these, 37,100 were classed as surface workers. From this it appears that the same general distribution of labor still is in effect as that found at the end of World War II. Slow improvement has been made, but during the month of September 1947, a large (for Japan) copper mine reported an underground force of 803 (34.6 pct) of a total labor force of 2321, not including staff technicians, while producing 12,000 tons of ore. Elimination of loans has forced the removal or shift of some surplus labor, although this is against Japanese traditional practice. The same copper mine cited above had 2420 employees classed labor as differentiated from staff in 1949, with 1042 underground (43.0 pct), with an average monthly output of 27,700 tons of ore.

Deconcentration

Another delaying factor in the recovery was the threat of reorganization hanging over most of the mining companies. The original policy established for the Supreme Commander for the Allied Powers to follow in dealing with the companies was to require major deconcentration and reorganization. With a clearer understanding of the limitations of

Table III. Nonferrous Metallurgical Plant Capacities by Company, In Pct of Total Capacity, 1948 Calendar Year*

Firm	Copper		Lead		Zinc	
	Smelter	Refinery	Smelter	Refinery	Electrolytic	Retort
Mitsui Mining Co.	5.0	7.3	29.0	37.0	51.0	100.0
Mitsubishi Mining Co.	20.6	18.6	34.0	21.0	30.0	
Nippon Mining Co.	30.8	20.8		31.0		
Nippon Soda Mining Co.			28.0	10.0	18.0	
Chugai Mining Co.			9.0	1.0		
Toho Zinc Co.		0.8				1.0
Furukawa Mining Co.	10.7					
Dowa Mining Co.	9.2	8.3		*		
Seika Mining Co.	19.7	22.8				
Dai Nippon Mining Co.	4.0					
Furukawa Electric Co.		21.4				
Total production, metric tons	120,940	115,560	21,100	42,725	29,020	26,000

* Source: company records.

‡ Michael B. Nicholson: Natural Resources Section Preliminary Study No. 14, Machinery Distribution in the Japanese Mining Industry (July 19, 1947).

§ Michael B. Nicholson: Materials and Power used in the Japanese Nonferrous Smelting and Refining Industry, Natural Resources Section Preliminary Study No. 32 (March 28, 1949).

Table IV. Subsidies and Loans to Metal Mining Industry[a]

Japanese Fiscal Year[b]	Subsidies: Gold Exploration[c]	Subsidies: Copper or Other Exploration[c]	Subsidies: Pyrite Price Adjustment[d]	Subsidies: Copper, Lead, Zinc Price Adjustment[d]	Government Loans: Equipment and Housing[e]	Government Loans: Operating Funds[f]	Government Loans: Counterpart Aid Funds[g]
1946[h]	2,000,000	None	66,038,000	None	19,000,000	345,179,000	None
1947[i]	4,200,000	None	None	801,414,000	374,001,000	796,421,000	None
1948	14,616,000	None	None	2,675,712,000	881,847,000	522,475,000	None
1949	14,440,000	59,337,200	None	None	None	None	308,000,000
1950	ND[j]	151,769,100[k]	None	None	None	None	80,000,000

[a] Source: Gold subsidy and loan data from Mining Bureau, Japanese Government and Mining Industry Association; price adjustment subsidy data from Bank of Japan.
[b] April of named year through March of following year.
[c] One half the cost of exploration (underground).
[d] Difference between consumers price and pool of average cost of three or more mines producing same mineral plus dealer's price and Kodan (Public Distribution Corp.) charge is paid as subsidy to the mining company.
[e] Includes a small amount for nonmetal mines.
[f] Includes funds for mining, milling, smelting, and refining.
[g] Matching yen fund based on amount of dollar support to Japan.
[h] July 1946 to March 1947.
[i] Includes some funds estimated on basis of budgeted items.
[j] Included with copper exploration.
[k] Includes gold subsidy of about Y14,000,000.

the industry, however, came relaxation of the original policy. Control of the mining industry actually had been fairly well balanced. The situation at the end of 1948, shown in Tables II and III, illustrates this point.

When the reorganization of the mining companies was finally directed on August 30, 1949, only Mitsui, Seika (Sumitomo), and Mitsubishi mining companies were affected. Even then the relatively good balance between the companies was recognized, so that these companies were required only to split into two companies, a coal, and a metal mining and metallurgical company. The separation of the coal and metal mining activities of the companies involved is logical because the coal mines are in no sense captive mines—the companies use but a minor part of the coal produced in their own metallurgical plants. In fact, Mitsui controlled about 16 pct, Mitsubishi 11 pct, and Seika about 4 pct of the national coal output for 1948. The break-up ordered should have no detrimental effect on the industry, and in the case of at least one concern improvement in operations has resulted.

Finances

General: With severe postwar inflation, which can be measured by the price of copper, which rose, as stated earlier, from a 1945 price of Y2,060 to Y181,060 a ton by 1948, rehabilitation and reconstruction work was slow in starting. Mine operators were reluctant to expend inflated yen. Reconstruction of several mills was delayed or suspended entirely, pending return to more or less normal conditions. In other instances, in mines formerly having large mills, the capacity of new plants was sharply curtailed; instead of a 6000 ton per month plant, burned in 1945, a copper mine has installed a 2500 ton per month plant. A gold-silver mine formerly having a 3000 ton per day mill capacity constructed a mill of 400 ton capacity.

Operating costs generally rose more rapidly than the controlled metal prices. Some of this difficulty was due to the fact that, while the mines found it necessary to obtain many items of supply through illegal and expensive channels, the cost of production on which metal prices were based was calculated by the pricing agency on the basis of official controlled prices of materials.

To provide funds for the mining industry to use in rebuilding and for meeting operating expenses, and in the absence of sufficient private capital, the Japanese Government took over the task of financing by subsidies and loans. During the same period every effort was made to hold metal prices down through both price and distribution controls.

Subsidies and Loans: Initially the primary needs of the industry were funds to pay operating expenses and meet payrolls. Thus in 1946, Y345,179,000 in loans were provided, largely from the Reconstruction Finance Bank of the Japanese Government, to be used to defray operating costs. A small amount was made available for housing and equipment. The sum of Y66,038,000 was paid to pyrite producers as a price adjustment subsidy to cover production costs, which were higher than the selling price.

By 1947 rehabilitation programs were developing rapidly and with them came the demand for funds in greater amounts. During this year loans of Y374,001,000 were provided for housing construction and equipment purchase and more than Y796,400,000 for operating expenses. Price adjustment subsidies were furnished copper, lead and zinc producers in the amount of Y801,000,000.

Government financing reached a peak in 1948 when more than Y4,000,000,000 in loans and subsidies were paid to mining companies. Action taken by the Supreme Commander for the Allied Powers ended most government loans to the mining industry in the fall of 1949. More money was allocated to exploration subsidies, however. The Nippon Mining Co. was granted loans from United States Aid Counterpart Funds for sulphuric acid plant construction at two smelters to utilize waste gases. The detailed breakdown of loans and subsidies provided to Japanese metal mining from 1946 to 1951 is given in Table IV. It should be remembered that the yen exchange rate has changed several times, so the subsidies and loans are not as formidable as they appear to be at first glance. From a pre-Surrender Y4.00:$1.00, the exchange rate has risen until the rate on April 23, 1949 was Y360.00:$1.00 at which point it has remained through July 1, 1951.

Although metal mining received considerable assistance, the coal mining industry was in a far more favored position than the metal during the postwar period, with subsidies and loans amounting to Y2,100,000,000 in 1946; Y19,300,000,000 in 1947, and Y33,300,000,000 in 1948, as compared to funds received by nonfuel mining industry of Y432,217,000

in 1946; Y1,976,036,000 in 1947, and Y4,064,650,000 in 1948.

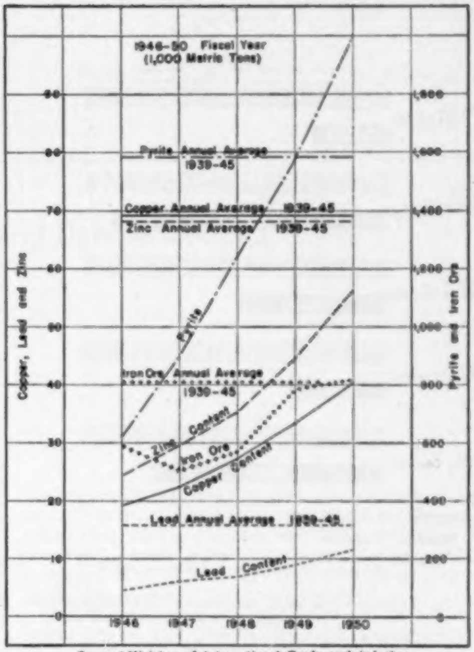
It should be noted that Japan's Government organization with respect to mining provides little in the way of the indirect assistance available in the United States. The Japanese Mining Bureau operates under the Resources Agency of the Ministry of International Trade and Industry and is almost exclusively an administrative organization. In 1948 the entire research budget amounted to only Y5,983,000 of the Mining Bureau's budget of Y850,000,000. Completely divorced from the other agencies concerned with underground resources is the Geological Survey under the Agency of Industrial Technology, Ministry of International Trade and Industry. The Survey has begun to provide some assistance to mine operators needing geologic advice or assistance. Peculiarities of the government mining organizations are that short time officials remain in any one position and the fact that the administrative officials do not remain in the same or related fields of activity. On August 1, 1951 the Chief of the Japanese Mining Bureau was transferred, after serving for 26 months, to the position of Chief, International Trade and Sundries Bureau (pulp, rubber goods, toys, etc.). On the same date the Chief, Mine Safety Bureau, was transferred after serving 5 months, to head the Iron and Steel Bureau (iron and steel production).

Controls and Prices: Although price controls were maintained after the end of World War II and price raises were made subject to the approval of the Occupation agencies, the general inflation brought about rapid price increases. The case of copper has been cited. Lead prices rose from Y1,800 a ton in 1945 to Y80,810 in 1948, zinc from Y2400 to Y58,036, pyrite from Y20.16 to Y2000 and iron ore from Y50 to Y1,154. By 1948 however, with increasing industrial production inflation had been brought to a virtual standstill, metal prices remained at the 1948 level through March 1949 and into 1950.

In conjunction with a general move to free the economy from unnecessary restrictions and governmental direction, many commodities which had been under both price and distribution control were released during September to October 1949. Among those released were copper, lead, zinc, and mercury; iron ore and pyrite controls were retained. As a result of the decontrol, the mining industry for the first time since 1938 was operating under relatively free economy.

Metal prices subsequent to decontrol reflected the production and stock situation rather well. Copper prices broke from Y181,060 because of large stocks derived from the smelting of scrap and curtailed spending by the Japanese National Railway, the largest copper consumer. Several large lots of copper were exported at prices well below world market in an effort to free capital tied up in copper. Mercury prices dropped as well, reflecting small demand and relatively large stocks. The demand for zinc remained strong through 1949 and 1950, resulting in a doubling in price. At the end of March 1951 pyrite and iron ore were decontrolled, and the prices of these items started a slow rise.

With the advent of the Korean war, metal stocks largely disappeared and metal prices moved upward rapidly. To illustrate the recent metal price history in Japan, metal prices and decontrol data are supplied in Table V. As commodity prices in general increased less rapidly, the mines are now enjoying a prosperous period.



Source: Ministry of International Trade and Industry

Fig. 1—Mine production, copper, lead, zinc, pyrite, and iron ore

Table V. Japanese Metal Prices and Price Decontrol Data, Yen Per Metric Tons[a]

Metal or Ore	July Metal Price, Yen Per Metric Ton 1949	1950	1951	Decontrol Data and Price Prevailing
Copper	181,000	170,000	300,000	Oct. 1, '49, @Y181,060[e]
Iron Ore[b]	1,298	1,298	2,600	Mar. 31, '51, @Y1,800
Lead	80,810	83,000	240,000	Sept. 2, '49, @Y80,810
Mercury	32,000[d]	20,000[d]	78,000[d]	Sept. 2, '49, @Y32,000[d]
Pyrite[c]	1,634	2,064	3,230	Mar. 25, '51, @Y2,585
Tin	520,000	570,000	1,500,000	Dec. 4, '48, @Y99,000
Zinc	58,030	116,000	250,000	Sept. 2, '49, @Y58,030

[a] Source: Japan Mining Industry Association.
[b] 50 pct Fe.
[c] 50 pct S.
[d] Price per 34.5 kg.
[e] Consumers price Y102,014.

Production

As measured by the increase in the output of mine products, the recovery of the Japanese metal mining industry since 1945 has been generally satisfactory. Metallurgical plant output, drawing on large scrap stocks and imported raw materials, has been nearly equal to demand and in some cases, prior to the outbreak of the Korean fighting in 1950, exportable sur- such as copper, lead and iron ore, being limited by pluses developed. Mine production of some metals such as copper, lead and iron ore, being limited by small reserves, has been less than demand.

Pyrite mining has made the most rapid recovery because of adequate ore reserves to support output and the demands of the fertilizer industry for more

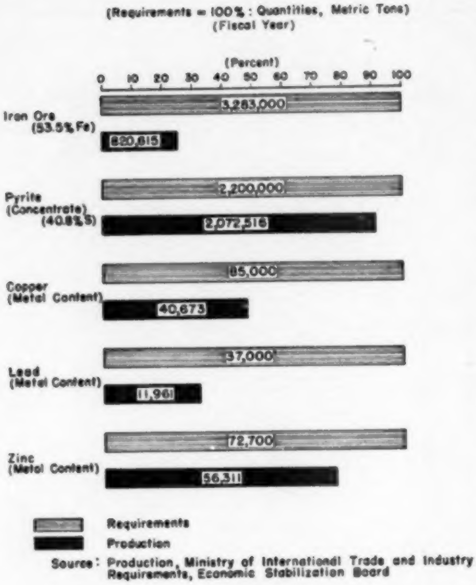


Fig. 2—Mineral and metal requirements 1951 compared with production 1950.

sulphuric acid. Fertilizer, one of Japan's most critical needs, has received particular attention from all agencies. Contributing industries, such as pyrite mining, have also come under the program. Most of the funds for equipment and housing, listed in Table IV, have been for pyrite mines or mines producing byproduct pyrite.

Mine Output: Between 1945 and 1950, mine production of major ferrous and nonferrous metals rose sharply; however production was far below the average for the 1939 to 1945 period, generally the period of maximum output. Pyrite production is the exception—having gone well above the 1939 to 1945 average. In Fig. 1 is charted the record of mine production 1946 to 1950.

Metallurgical plant output for 1950, when compared with mine production figures, illustrates the role of scrap and imported ores and concentrates in providing Japan with an adequate supply of metal, see Table VI. Since scrap stocks have been in large measure eliminated, Japanese metallurgical plant operators are entering the world market in the search for concentrates.

Table VI. Production of Metal in Concentrates Compared with Refined Metal Output, 1950,[a] Metric Tons[b]

Form	Copper	Lead	Zinc	Iron
Refined metal	89,690	17,211	52,481[c]	1,935,500[e]
Metal in concentrates	40,673	11,961	56,311	434,926
Total from scrap or imported concentrates and ore	49,017	5,250	6,170[d]	1,500,774

[a] Japanese Fiscal Year.
[b] Source: Ministry of International Trade and Industry.
[c] Electrolytic, 35,147; distilled, 17,334.
[d] Surplus.
[e] Pig iron.

Adequacy of Supply: In Fig. 2 a comparison is made between the output of the metal mines for 1950 and the estimated requirements for 1951 to indicate something of the degree to which the needs of the country are being met. Unquestionably pyrite requirements can be supplied, and an increase in zinc production of over 20 pct is possible in view of the strenuous efforts being made to add to plant capacity. With scrap and imported ores, copper needs possibly may be met without imports of refined metal, but lead metal imports unquestionably will be needed as will imports of iron ore.

Japan's metal mining industry undoubtedly will continue to play an important role in her economy. However, the part played by any given segment will be controlled by the ore reserve situation. Japan has reserves sufficient to support production of pyrite for her own needs and perhaps for export, and with added exploration and development, zinc probably can join pyrite. At high cost, most of Japan's copper needs can be satisfied but lead and particularly iron ore are limited in amount. Nonmetallics used in the processing of metal ores, such as limestone and refractory silica and clays except special-purpose high-aluminous clays, are plentiful. Graphite and manganese, although not high grade, are relatively plentiful but not adequate to meet demand. High grade chrome is available in limited amounts, as is refractory chrome. One encouraging feature in the reserve picture is the tremendous improvement in the quantity and quality of geologic exploration work by the mining companies. Little tungsten is available. An important shortage, particularly from the standpoint of the iron and steel industry, is that of suitable coking coal. Although good metallurgical coke can be made from Japanese coals in adequate quantity by proper blending and by use of additives of various types, the use of such coke poses economic and operating problems which are of sufficient magnitude that coking coal will be imported if available, even at high cost.

Occupation Headquarters

After the surrender of Japan, technological assistance to the Japanese metal mining industry was provided by the Mining and Geology Div., Natural Resources Section, General Headquarters, Supreme Commander for the Allied Powers, General MacArthur's Headquarters in Tokyo. The Division organization roughly paralleled that of the U. S. Bureau of Mines, with a Minerals Branch, Metallurgy Branch, Solid Fuels Branch, Petroleum Branch, and a Minerals Economics Branch. In addition to the technical assistance program carried on by the Division, studies were made of the various mineral commodities to provide background information for planning. A series of reports covering the results of the studies have been issued. Through July 1951, 75 reports and supplements and 21 preliminary studies, covering the reserves, mining, and milling of ore petroleum reserves and production, and metallurgy were published. In addition, five special studies have been made covering machinery distribution, administration of the mining industry or mining methods. The reports are available in photostat or microfilm from the Office of Technical Services, U. S. Department of Commerce, Washington 25, D. C. The issuing agency should be cited as Natural Resources Section, General Headquarters, Supreme Commander for the Allied Powers.

Collector Mobility and Bubble Contact

by M. D. Hassialis and C. G. Myers

THE nature of a collector-coated mineral surface has been the subject of some experimentation and much speculation. Various aspects of the problem have been isolated and studied; it is probable, however, that there exist aspects as yet unrecognized, which are needed for a complete description of the system. The recognized elements of the problem are: 1—The nature of the chemical units comprising the film and the nature of the forces holding these chemical units to the mineral surface; 2—The orientation of the chemical units at the surface; and 3—The disposition of the collector units at the mineral surface. This report touches upon all three aspects of the broad problem; it deals, however, with a new and heretofore unrecognized element—the mobility of the collector units in the two dimensions of the solid-liquid interface.

Regardless of the viewpoint adopted as to whether the chemical units comprising the collector film are collector ions or molecules, held either by chemical or physical forces, the writers on flotation theory are in apparent agreement as to the orientation of the chemical units at the mineral surface. Wark and Cox[1] showed that the contact angle at a stable collector-coated surface in a constant chemical environment is independent of the nature of the mineral and of any resurfacing agent; it depends solely upon the collector used. They showed further that the value of the contact angle increases with the number of carbon atoms in a homologous series of compounds. Taggart, Taylor, and Knoll[2] postulated that the collector ion is oriented with the chemically reactive, polar end toward the mineral surface and the nonpolar, hydrocarbon-like end toward the liquid phase. In support thereof they cited the abstraction of p-p[1] dihydroxydiphenylthiourea and of glycol xanthate[3] by galena without a concomitant increase in the contact angle above that of clean galena. It was argued that in these cases the water-repellent layer which normally would be presented to the liquid phase owing to the orientation of the collector ions is overlain by a layer of water-avid groups, the hydroxyl groups. Implicit is the analogous orientation exhibited by a monomolecular layer of an insoluble substance at an air-water interface. It should be pointed out that these facts also are consistent with the concept that the collector units be flat on the surface of the mineral and that the response of the surface to an air bubble is determined by the ratio of the *area covered by water-avid groups* to the *area covered by water-repellent groups;* the areas being weighted according to the relative water-avidity or water-repellency of the groups involved.

The disposition of the chemical units of the collector film at the mineral surface has not received the attention it merits. Taggart[4] concluded from the observed stoichiometry of the ions entering and leaving a surface that the stoichiometry is obeyed in detail at the surface of the mineral. Thus, in the case of xanthate-coated galena, pairs of xanthate ions are attached to surface lead ions. In the case of *chemisorption,* Gaudin and Preller postulated an agreement between the atoms or ions of the substratum (the mineral surface) with those of the superstratum (collector film); in the case of the physical adsorption their position is not clear. In one place they state "the first type (physical adsorption) does not require any agreement between the atoms or ions of the substratum with that of the superstratum" and in another place that "It seems, in fact, as though some slight influence of the substratum is formed even in physical adsorption." The present work shows that in some cases of chemisorption the chemical units of the collector film are in constant surface-bound motion and may not be assigned to any site of the surface, hence there is no permanent association between the chemical units of the film and the atoms or ions of the solid.

The physical property of the collector film that is of paramount importance is its surface energy. Although this property is not susceptible to direct experimental determination, the related contact angle is. Much of the knowledge of the properties of a collector film derives (by deductive and inductive reasoning) from the behavior of an air bubble thereat

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This paper represents part of a thesis by C. G. Myers, submitted in partial fulfillment of the requirements for the degree of Doctor of Philosophy in the Faculty of Pure Science, Columbia University.

as determined by the contact angle. This knowledge, however, deals with the equilibrium state wherein contact has or has not been made and it does not deal with the transitory states through which the system passes in the attainment of equilibrium. From the practical viewpoint, the latter information may be and probably is the more important knowledge for it is highly improbable that true equilibria are ever established in the complex physicochemical system, which is the operating flotation machine.

To illustrate this, consider a volume of liquid in which a collector-coated particle and an air bubble are immersed. Knowledge of the surface energy of the liquid and of the contact angle is sufficient to calculate the change in the free energy of the system[6] between its initial state and the final state wherein the bubble is attached to the particle. The sign of this change indicates the direction of the trend towards equilibrium and specifies the equilibrium state but yields no information as to the rate and mechanism of the attainment of equilibrium (contact). It is not unusual to find that a system in a state of unstable equilibrium requires an additional push or activation or catalyzing action before it will move towards its equilibrium state. For example, a mixture of two volumes of hydrogen and one of oxygen is highly unstable but will remain as such for an infinitely long time. Yet when activated by an infinitesimal amount of energy in the form of a spark or by a catalyst, it goes to water with explosive violence. This additional push, this activation energy is the key to many successful commercial processes; knowledge thereof derives from a detailed study of mechanism.

The importance of induction time (bubble-contact time) was noted by Sven-Nilsson[7] and by Taggart and Hassialis.[8] The induction time appears to be related to the rate of attainment of the equilibrium contact angle, but the relationship is obscured by the unknown effect of the undetermined and variable compression of the bubble against the solid surface during the induction time. No explanation of this phenomenon has been offered previously. The present authors consider the mechanism of induction time to be as follows: The presence of an air bubble compressed against the solid surface creates a field of force in the intervening region that acts to restrain chemical units of the collector film which attempt to leave the region in the course of their surface-bound motion. Chemical units outside this region that enter in the course of their random motion are similarly restrained. With time this results in a net increase of surface concentration of collector. When this concentration exceeds some critical value, the pressure of the gas in the bubble is sufficient to disjoin the intervening liquid layer and establish bubble contact with the collector-coated surface.

The galena particles used in the experimentation were cut with an alundum disk from large, naturally occurring, single crystals. One face of each particle, referred to as the working face, was a face of the original crystal. The particles were ground with water on successively finer Norton emery polishing papers finishing with No. 4/0. The edges and corners were similarly prepared. The particles were polished with levigated alumina on a glass lap covered with cleaned and denapped Buehler "Selvyt" polishing cloth. Preparation was considered complete when all faces showed no scratches at 150X magnification. Final dimensions of particle No. 1 were 13x9x8.5 mm; of No. 2, 14x14x6.5 mm.

The alumina used in polishing was prepared from hydrated alumina No. C-730 purchased from the Aluminum Ore Co. This was heated at 1000°C for 1 to 2 hr and then levigated in distilled water for 15 to 30 min as desired. The unsettled alumina was siphoned off and concentrated by settling at a pH+7 to a slurry of about 10 pct solids by weight. The slurry was bottled in 250-g batches. All glasswear used was either Pyrex or Nonsol. Contamination by dust or organic matter was watched for and rigidly excluded. The polishing cloths finally adopted were the cotton-base type; no wool-base cloth was obtained which with or without treatment would not contaminate the particles. The polishing cloths were denapped by polishing a rejected particle for about ½ hr using moderate pressures. The cloths were then boiled for several hours in dilute aqueous sodium carbonate, rinsed with distilled water and then allowed to stand for about ½ hr in very dilute aqueous hydrochloric acid. Finally the cloths were rinsed with distilled water and allowed to stand in several changes of distilled water until used.

Particle handling was done with rubber gloves. Prior to use the gloves were dipped in cleaning mixture, rinsed with distilled water, and allowed to stand in distilled water to desorb any remaining traces of acid. The gloves were rinsed again before and after the operator put them on.

All glassware used in the work was Pyrex. The distilled water storage bottles, though not Pyrex, were made of low-solubility glass. Glassware was thoroughly cleaned with cleaning mixture, rinsed with distilled water, soaked with distilled water and finally rinsed prior to use.

Cleaning of the particles was performed on a glass lap covered with cleaned "Selvyt" cloth. The abrasive was fed to the lap from a bottle equipped with a dropper. Initially, copious amounts of abrasive are used; as cleaning proceeds, the volume of abrasive is reduced rapidly until finally only water is used. This being continued until no abrasive is visible on the lap. In this manner surfaces free of slime visible under 500X magnification are readily produced.

The criteria of particle cleanliness established for this work were the absence of a contact angle after an induction time which was longer than that used in the test and the development of a full contact angle to a 1-min induction time after conditioning with 30 mg per liter potassium ethyl xanthate solution. This last test was run at the end of an experiment to show that depression had not taken place during the course of the experiment.

The xanthates used in the experimentations were purified by the methods described in a previous publication.[9]

Mobility of Collector Ions

The results reported herein are the average results of a number of repeat experiments; the experimental procedure being fixed to give reproducibility.

Experiment No. 1—A galena particle was cleaned on all six faces and tested for cleanliness on its working face and one adjacent side by a 1-min contact with an air bubble in distilled water. Failure to develop any cling was taken as evidence of cleanliness. The particle was then conditioned for 1 min in a 30 mg per liter KEtX solution after which it was washed with distilled water and tested in distilled water with an air bubble. Both the working face and adjacent side showed a 60° angle after a 1-min induction time. The working face of the

particle was then polished without abrasive, under a continuous stream of water using a pressure which was insufficient to leave a galena streak on the cloth. After 5 min of such cleaning the working face tested 60° after a 1-min induction.

Experiment No. 2—The preceding experiment was repeated with the sole modification that the pressure exerted on the particle when the xanthated working face was being cleaned was sufficient to leave a galena streak on the cloth. The working face and the adjacent side both showed a very slight cling after a 1-min induction. This test was run either as a continuation of Experiment No. 1 or on a freshly prepared particle; the results were the same. Absence of depression was indicated by the fact that particles at the end of Experiment No. 2 developed the normal contact angle after conditioning in xanthate solution.

Experiment No. 3—A galena particle was cleaned until it tested 0° after 1-min induction on all six sides. It was conditioned for 1 min in xanthate solution and all sides tested with an air bubble after a 1-min induction time with the result that all sides showed a 60°, ± 3°, contact angle. The working face was then polished using levigated alumina for perhaps 10 sec, followed by polishing under a flood of water for some 4 to 5 min. The working face and one adjacent side showed 0° after 1-min induction, the remaining faces showed barely detectable clings. Subsequent conditioning of the particle in xanthate solution produced the normal angle on all faces. A blank test was run omitting only the actual abrasion (even though alumina slurry was splashed on the specimen); there was no reduction in the contact angle of any of the faces.

The reduction in contact angle of the sides tested is related to abrasion of the working face, whether this abrasion is produced with or without alumina.

That it is not related to any other part of the test procedure is indicated by Experiment No. 1 and the blank. That it is not depression caused by alumina slimes or by some soluble chemical is indicated by the production of the normal angle upon retreatment with xanthate solution. The reduction in contact angle indicates therefore removal of the collector film that is responsible for the contact angle. Removal of the collector film does not take place by dissolution or chemical destruction. This is shown by Experiment No. 1, by the blank, and by an auxiliary experiment in which a xanthated galena particle was washed with some 40 liters of distilled water over a 3-hr period without any noticeable reduction in contact angle. This auxiliary test also shows that the film is not removed by diffusion into the body of the solid. It follows that the only way in which the collector film could have left the surface is via the galena streak laid down on the polishing cloth. Abrasion of the working face produces a large number of extremely small galena particles having an aggregate surface area, which is large compared to the surface area of the particle. For the collector film to leave the specimen with these minute particles, it is prerequisite that the chemical units of the film must move from all other sides of the specimen to the side being abraded and thence to the particles produced by abrasion. To test this point the following experiments were performed.

Experiment No. 4—Galena particle No. 2 was cleaned until it showed a 0° contact angle after a 1-min induction period on all sides and a 0° contact angle after a 45-min induction time on its working face. It was then conditioned for 1 min in a 30 mg per liter ethyl xanthate solution, after which it showed a 60° contact for a 1-min induction time on its working face. Galena particle No. 1 was cleaned to a 0° angle on its working face for a 1-hr induction period, and a 0° angle after a 1-min induction time on all other faces. Both particles were transferred to a cell filled with distilled water and so arranged therein that their working faces were vertically disposed, parallel to each other and separated by the smallest possible distance just short of actual contact. The system was covered and left untouched for 20 hr, 12 min after which time the particles were removed and tested. Particle No. 2 showed a 60° angle after a 1-min induction time on its working face; particle No. 1 showed a 0° angle after the same induction time on its working face. Particle No. 1 was then conditioned in xanthate solution and developed the normal xanthate angle.

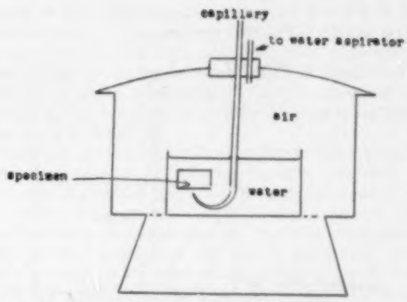

Fig. 1—Equipment for multiple bubble-contact experiments.

Experiment No. 5—The preceding experiment was repeated with particles of like cleanliness. When the xanthated and unxanthated particles were transferred to the water-filled cell they were arranged so that one particle rested upon the other with their working faces in contact. After 20 min of such contact the particles were removed and their working faces tested. The xanthated particle showed a 60° angle and the unxanthated particle a medium cling, both after a 1-min induction time. They were then returned to the cell and arranged as before. After a total contact time of 17 hr, 13 min the particles were again removed and the working faces tested. The xanthated particle showed a 51° angle and the unxanthated particle a 42° angle, both to a 1-min induction time.

The first of these tests shows that collector film is unable to migrate across a water gap from a coated to an uncoated particle. This might have been deduced from the auxiliary test cited above. These statements should not be interpreted as denial of partition of the collector film between the contiguous liquid and solid phases, but that the partition equilibrium is so far in the direction of the solid as to make negligible the effects of the collector partitioned in the liquid. The second test shows that when physical contact is made between specimens, even though this contact probably consists of a limited number of point contacts, collector film migrates from the coated particle over the solid-to-solid bridges to the unfilmed particle. There appears to be no other way in which this could have happened. Once again, mo-

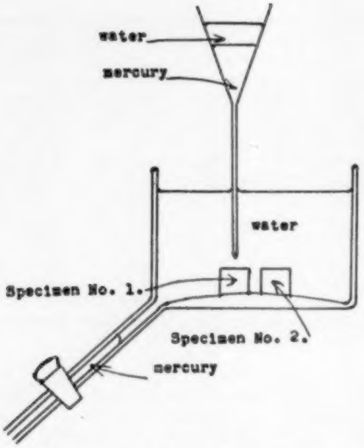

Fig. 2—Equipment for mercury-droplet experiment.

bility of the chemical entities of the collector film in the solid-liquid interface must be presupposed to explain the facts or the facts may be used to deduce the mobility.

Another interesting aspect of Experiment No. 5 is that the amount of xanthate film transferred from particle No. 2 to particle No. 1 produces an increment in contact angle of the latter particle of 42° and a decrement in contact angle of the former particle of only 9°. Even if allowance is made for the difference in surface areas (756 sq mm for No. 2 and 608 sq mm for No. 1) of the two particles, it follows that equal increment or decrement of collector to or from a surface will not produce equal increments or decrements in contact angle. This means that the relation between contact angle and surface coverage or concentration is not linear. It can be further deduced that the rate of change of contact angle with surface concentration is large for small concentrations, decreases with increasing concentration and finally vanishes after some characteristic concentration is exceeded.

The mobility of molecules in a gas-liquid interface[9] and of molecules and atoms in a gas-solid interface[10-13] has been established by many independent experiments. It is curious that no experimental evidence has been advanced in support of similar mobility in the solid-liquid interface. The following group of experiments was designed to determine whether the chemical units of the collector film have the ability to move into and in the gas-liquid interface.

Experiment No. 6—Particle No. 2 was cleaned until it tested 0° after a 1-min induction time. It was then conditioned for 1 min in a 30 mg per liter potassium ethyl xanthate solution, then washed and transferred to a cell filled with distilled water. Particle No. 1 was cleaned until it tested 0° on its working face after a 26-min induction time. It was then placed in the water-filled cell next to but not touching particle No. 2. Both particles were arranged with their working faces horizontal and facing upward. A captive bubble was compressed against the working face of the xanthated particle for 1 min, then removed and transferred to the working face of the unxanthated particle against which it was compressed and maintained for 1 min, finally the bubble was removed and liberated. This cycle was repeated with a fresh captive bubble 150 times. The contact angles for a 1-min induction time were determined for the xanthated and unxanthated particles; at the beginning of the experiment the values were 60° and 0° respectively, at the end of the experiment the respective values were unchanged. By measuring the diameter of the gas-solid contact area and the maximum diameter of the bubble as viewed in the ground glass of the camera of the normal setup, it was possible to estimate the total gas-liquid area and the total gas-solid area; these were about five and two and one-half times the surface area of particle No. 2. At the end of the experiment the unxanthated particle was tested for depression as in previous experiments and none was found.

Experiment No. 7—Particle No. 1 was cleaned until its working face tested 0° after a 5-min induction time. It was transferred to a cell filled with distilled water and there placed with its working face down on a glass tripod. The cell was placed in a vacuum dissicator and arranged as shown in Fig. 1. When the aspirator was turned on, air bubbles released from the tip of the capillary tube rose and collided with the working face, after rebound from this initial collision they proceeded by a succession of bounces to a vertical side of the particle and then escaped to the free water surface where they broke. The bubble rate was determined to be approximately 200 bubbles per min. The projected area of 30 bubbles was determined approximately as 6.1 cm^2. The experiment was run for 3 hr, 7 min, at the end of which time the working face of the specimen was tested and found to have a 0° angle after a 1-min induction time. Depression was tested for and found to be absent.

Experiment No. 8—Particle No. 1 was cleaned to a 0° angle after 1 min in a 30 mg per liter xanthate solution, after which treatment the contact angle was determined to be 60° for a 1-min induction time. Experiment No. 7 was then repeated with this particle using a 60 bubble per min rate, the time of the run being 19¾ hr. Bubble behavior was of three distinct types: 1—Travel across the working face by a series of bounces of progressively decreasing amplitude; 2—rolling along the surface without any visible bouncing, this being the predominant behavior; and 3—rolling, then sticking to the surface until their volume had been increased sufficiently by coalescence, owing to collision with subsequent rolling and bouncing bubbles, to permit dislodgement and escape. It was estimated that the projected area of 60 bubbles equalled the area of the specimen. At the end of the run the working face tested 60° after a 1-min induction time.

It may be concluded from Experiments 6 and 8 that any motion of the entities of the collector film into the gas-liquid interface or partitioning therewith is so small as to be undetected either by a decrement in contact angle of the source particle or by an increment in the contact angle of the unxanthated particle. Experiment 7 is really a blank run, proving that contamination introduced by the air is insufficient to contaminate a clean particle. Without it, interpretation of Experiment 8 would be obscured by the possibility that collector film lost by partitioning in the gas-liquid interface was replaced by air contamination sufficient to maintain the contact angle. The results of Experiment 7 are con-

sistent with the observation that a freshly swept gas-liquid interface takes several minutes exposure to laboratory air before contamination is detectable. It should be noted that the total projected area of the colliding bubbles in Experiment 8 was 1200 times the surface area of the particle. This large gas-liquid area was provided purposely to compensate for a possible difference in ability to transfer under the dynamic conditions of this experiment as compared to the relatively static conditions of Experiment 6.

Mobility of the chemical units of the film in the gas-solid interface was not investigated; mobility across a bridge formed of two dissimilar solids was approached by the following controlled experiment.

Experiment No. 9—Both galena particles were cleaned until they tested 0° after a 1-min induction time on all of their sides. They were then conditioned for 1 min in 30 mg per liter xanthate solution and tested to insure that the normal xanthate angle had developed. Both particles were transferred to the apparatus shown in Fig. 2 and arranged with their working faces up but with neither face directly under the fine capillary tip of the funnel. Chemically cleaned and redistilled mercury,[14] which had been stored under distilled water, was poured into the funnel. The mercury droplets effluxing from the tip of the funnel fell through the water in the cell and collected in the leg. After several minutes the cell was moved relative to the independently supported funnel so that the mercury droplets impinged on the working surface of particle No. 1. The mercury previously collected in the leg was drawn off by means of the water-lubricated stopcock. The rate of dropping was 240 droplets per min. Initially no rebound of the droplets from the working face was observed; they rolled along a relatively narrow path to an edge and then dropped to the bottom of the cell where they collected in the annular depression shown in Fig. 2 and finally in the down leg. Contact between the droplets and the underside of the particles was prevented by seating the particles on the central raised portion of the cell floor. The level of the mercury in the leg was never allowed to rise to the level of the cell bottom, this was achieved by periodic draining of the mercury into a water-filled flask. After 49 min of running time, the cell was shifted so that no mercury dropped on either particle. The contact angles of the two particles were determined in situ and found to be a slight cling for No. 1 and 60° for No. 2, both after a 1-min induction time. The cell was then shifted to bring particle No. 1 under the dropping mercury and the run continued. After some 15 min elapsed time, it was observed that the droplets were rebounding vigorously after their initial collision and were proceeding to the discharge point by a succession of rebounds of decreasing amplitude. When 24 min had elapsed, the experiment was discontinued. The working face and its opposite side of particle No. 1 both tested 0° after a 1-min induction time. Depression was tested for and found absent. The working face of particle No. 2 showed 60° after a 1-min induction time. The mercury which had been collected tested 49° after a 1-min induction time whereas initially it had shown only a light cling for the same induction period. When the collected mercury (260 gm) is poured from one flask into another, always under water, the freshly created surface shows only a strong cling after a 1-min induction time and 46° after a 5-min induction time. The surface of the mercury in the leg during the course of the experiment appeared gray and powdery. Fresh mercury was poured into a cell filled with a 30 mg per liter xanthate solution; after about 1 min most of the xanthate solution was removed, and the mercury surface showed a 61° angle after a 1-min induction time and 70 to 75° for the same induction time 15 min later.

Mobility of the collector film from the xanthated galena particle to the mercury is shown by both the decrease in contact angle of the galena particle and the increase in contact angle of the mercury. That depression is not responsible for the decrease in angle of the xanthated particle is indicated by the maintenance of its contact angle by particle No. 2 and by the ability of No. 1 to regain the normal xanthate angle after retreatment with xanthate solution. The behavior of the droplets at the surface of the particles indicates that in the early part of the experiment mercury-solid contact was made; whether the interface is mercury-oriented xanthate or mercury-galena is not known. It is probably the former, otherwise the bouncing noted in the latter part of the experiment and which indicates the absence of a force restraining droplets to the surface should not have taken place when the surface was freed of xanthate film.

Although mercury has many metallic properties, the preceding experiment does not establish migration across a bridge made of two dissimilar solids. A single preliminary test of the type performed in Experiment No. 5 was made using a copper block cleaned to a 0° angle after a 1-min induction time and a xanthated galena particle. After 15-min contact time the copper block tested a very strong cling (just barely immeasurable) and the xanthated particle 58° both for a 1-min induction time.

Up to this point, reference has been made to the mobility of the chemical units of the collector film. It is pertinent to ask: Are these chemical units xanthate ions or lead xanthate molecules? No definite answer can be given to this question based upon the facts presented. The gray powdery appearance of the surface of the mercury collected in Experiment 9 indicates the existence thereat of an insoluble solid phase, which may be either mercury or lead xanthate. Only in the latter case would argument be advanced to support the thesis that lead xanthate migrated. In the former case, either ion-migration followed by reaction with mercury or lead-xanthate migration followed by reaction with mercury could have occurred.

The time required for the ions or molecules to transfer in the various experiments is unexpectedly long. If it be assumed that the ionic or molecular velocities are comparable in magnitude to those obtaining in liquids or gases, then a fraction of a minute should have sufficed in most cases. If it is assumed that there are bound to the interface other ions or molecules (hydrogen ions, hydroxyl ions, water molecules) and that these also possess mobility in the interface, then the migration of the xanthate ions or molecules is opposed by the counter migration of these other ions or molecules. The resulting collisions, with the consequent shortening of the path between collisions and the changes in direction of motion, will reduce the overall velocity of migration to such an order of magnitude as is associated with diffusional processes; self-diffusion is, of course, always possible. Experiments reported elsewhere[10] indicate that the rate of transfer is not inversely proportional to the square root of the mass of the

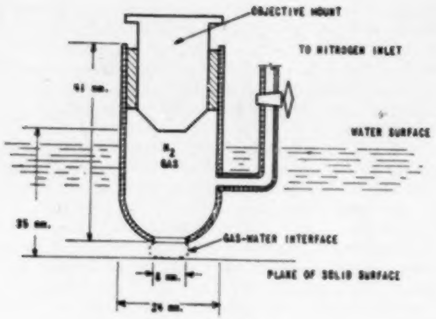

Fig. 3—Bubble holder.

migrating ion as might be expected from kinetic theory.[19] Thus the ratio of the rates of transfer in comparable experiments for ethyl and hexyl xanthate should be inversely proportional to the square root of the ratio of the masses, that is, proportional to 1.21. The experimental ratio is much greater. The migration velocity also possesses an unusually high temperature coefficient; thus a decrease of 25°C from room temperature to 0°C is sufficient to slow the migration of ethyl xanthate ions on galena to a barely detectable rate.

It is too early yet to predict how this new phenomenon will be used in flotation practice; it can be predicted on general principles, however, that insofar as it puts in the hands of the operator another control of the flotation process it is bound to be useful. To illustrate possible utility consider the following example, realizing that it is purely speculative and without any experimental justification: It is desired to separate the molybdenite content of a copper concentrate. Assume that the collector used in the production of the concentrate exhibited mobility or that the temperature of the concentrate is raised to enhance mobility. A small amount of a powder of very high specific surface is mixed with the concentrate and intimate contact produced by stirring. The collector ions migrate from the concentrate to the surface of the powder. The powder is then separated by simple classification or floated off with molybdenite after the addition of neutral oil as in the normal procedure.

Induction-time Phenomenon

It has been shown[9] that for a fixed time of conditioning in a xanthate solution the induction time required to develop the full 60° angle increases as the concentration of the xanthate solution decreases. It was also shown that if, after development of the full angle, the bubble were withdrawn and then immediately returned to and compressed against the same part of the mineral surface, approximately the same induction time was required to redevelop the full angle.

The pressure in excess of the hydrostatic pressure required to maintain an element of area of the bubble in equilibrium is $T\left(\frac{1}{R_1}+\frac{1}{R_2}\right)$,[21] where T is the surface tension and R_1, R_2 are the principal radii of curvature. Since the gas pressure inside a bubble must be uniform, the only way in which elemental areas at different hydrostatic levels can remain in simultaneous equilibrium is for the areas to adjust their curvature, that is, by varying R_1 and R_2. An uncompressed captive bubble is in equilibrium under the combined actions of gas pressure, hydrostatic pressure and curvature. Upon compression the bottom of the bubble is flattened out, that is, R_1 and R_2 are increased (in general by a multiple of the original value). It follows that $T\left(\frac{1}{R_1}+\frac{1}{R_2}\right)$ is considerably decreased, the gas pressure inside the bubble is too large for this area of the bubble. What maintains this area in equilibrium is the reaction of the intervening liquid layer backed by the solid. Since the gas pressure is in excess of the hydrostatic pressure at this point, it follows that any liquid intervening between the bubble and the surface should be displaced. It is not displaced when the solid surface is clean, hence the liquid is held against the solid surface by a force which in combination with the hydrostatic pressure exceeds the gas pressure. It also follows that when the intervening liquid is displaced (as in the case of a collector-coated surface) this force in combination with the hydrostatic pressure is less than the bubble pressure. Since under identical circumstances the gas pressure within a small bubble is larger than the pressure in a large bubble, easier displacement of the intervening liquid layer by a small bubble might be expected. This accounts for the often observed fact that it is easier to obtain contact with a smaller captive bubble.

The induction time cannot be explained by appealing to the viscous resistance of the liquid which would oppose flow of liquid out of the intervening zone. Viscous resistance is proportional to velocity, hence it vanishes at vanishing velocities. Simple consideration of the velocity with which the infinitesimal volume of liquid in the intervening zone must move to require an induction time of say 10 min, renders viscous resistance an improbable cause. The same consideration convinces one that when the bubble exceeds the sum of the hydrostatic and adsorption pressures, the intervening film should move out of the compression zone quite rapidly.

The key to the problem appears to be the change with time in the magnitude of the force holding the intervening liquid layer. This force is manifested in many other phenomena normally classified as adsorption phenomena.[18] It decreases in magnitude as the solid surface becomes coated with a liquid-repellent film.[20] If, therefore, water repellency of the surface can change with time, a probable mechanism is found. The first part of this paper presented evidence proving the mobility of collector ions in the solid-liquid interface. Collector ions located on that part of the solid surface against which the bubble is compressed and in closest contact with the bubble owing to solid surface irregularities are acted on by a force of attraction directed toward and owing to the solid (which force constrains these ions to move in the surface) and a force of attraction[20] directed toward and owing to the gas molecules. These forces have no component in the direction of motion of the ion until the ion, in the course of its random motion, reaches that part of the surface above which the bubble wall is not parallel thereto. At this point the attractive force of the gas molecules has a surface-parallel component directed

towards the center of the compression zone. This component acts to restrain ions from moving out of this zone. Ions outside of the compression zone, in the course of their random motion, will enter the compression zone. The net result is in an accumulation of collector ions with time in the compression zone. As the number of these ions per unit area increases, the adsorption force acting on the intervening liquid decreases. When some critical surface concentration of ions is attained, the adsorption force will have been sufficiently decreased that its resultant with the hydrostatic pressure is less than the bubble pressure, and the intervening liquid layer is displaced, that is, gas-solid contact is established.

When the gas-solid contact is broken by withdrawal of the bubble, the collector ions move out of the area of high concentration, and conditions for establishment of contact are destroyed. It was previously noted that the rate of transfer of collector ions is apparently slow; here the evidence indicates that the movement out of the area of concentration is rapid because the usual time elapsed between withdrawal and recompression of the bubble is of the order of 1 to 3 sec. Two factors should be considered in this connection. The surface-concentration gradient existing between the area of high collector-ion concentration and the rest of the surface is high, and change of concentration should therefore be correspondingly high when the restraining effect of the bubble is removed. Even though the change in this concentration in 1 to 3 sec is not great, the time required to reattain the critical concentration when the action of the bubble is brought to bear may be long for in diffusional processes the equilibrium concentration is reached in a manner asymptotic to a line parallel to the time axis. It is not unusual to find that 95 pct of the change in a diffusion process takes place in the first minute, while the remaining 5 pct requires days. The combined action of these two factors may suffice to explain the apparent contradiction noted above.

Another factor bearing on this point is the factor of orientation. As previously noted, most investigators tacitly assume orientation of the collector ions; it was also noted that this condition is not necessary though it may be sufficient. The assumed orientation is open to question on two counts; first, that the potential energy of an isolated xanthate ion on the surface of galena is a minimum when it is lying flat on the surface,[8] second, that there exists within the xanthate ion a second point of attraction for the surface, the oxygen of the ether bond. The flat position appears more probable for an isolated xanthate ion. When the ion is not isolated it may be reoriented to the position normally postulated under the action of either a lateral force of attraction between xanthate ions or a lateral pressure owing to momentum transferred by colliding xanthate ions. Such reorientation would have a marked effect upon the magnitude of the adsorption force acting on the layer of water intervening between solid surface and bubble because it increases the distance over which the force of attraction, exerted by the galena surface upon the water molecules of the intervening layer, would act.

To obtain a clearer understanding of the induction-time phenomenon and of the mechanics of bubble contact, an experiment was devised permitting microscopic examination of the solid surface immediately below the compressed bubble.

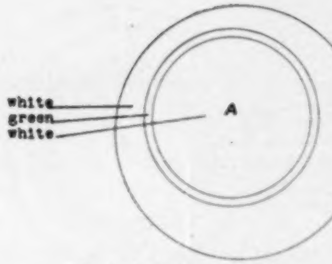

Fig. 4—Initial interference rings.

The microscope used was a Leitz, equipped with a Beck vertical illuminator, a 40-mm objective and a 20x Periplan eyepiece. Total magnification was 64x, working distance 35 mm. A small glass cup, provided with a side arm and opening was attached to the objective as shown in Fig. 3. The cup was cleaned by racking the microscope tube downward until the lower edge of the cup was about 30 mm below the free surface of cleaning solution contained in a cell placed on the stage. As the stopcock was closed during this operation, the cleaning-solution level rose within the cup to a height of only 15 mm above the bottom. Cleaning solution was removed by rinsing repeatedly with distilled water using the same procedure.

Nitrogen cleaned by bubbling through sulphuric acid followed by water is admitted to the cup immersed in a cell filled with distilled water. The gas is turned off and its pressure in the cup adjusted by bleeding so that the distance from the bottom of the bubble formed at the lower edge of the cup to the front lens of the objective exceeded the working distance of the objective by 1 to 3 mm. A clean galena particle placed in the cell was brought into position under the bubble and the stage racked upwards until bubble compression was effected and the surface of the galena came into focus. Within a matter of seconds a narrow green ring appeared in the field of view as shown in Fig. 4. This was followed by a complex sequence of changes within the next minute or two. The original green ring changed color and broadened, simultaneously concentric color rings appeared inside and outside of the original ring. The color cycle of the original green ring went from green to its first red, the first interior concentric ring appeared as green. It then followed a color cycle similar to the initial ring ending with red. While the first interior ring was changing from green to its first red, a second interior ring appeared within the first as green. This underwent a similar color cycle ending with bright blue. While the second interior ring was changing to its first red, a third interior ring appeared as green; it went through a similar color cycle ending with green. While the third interior ring changed to its first red, a fourth interior ring appeared as green, and an exterior (to the initial green) ring appeared as green. The appearance of the interference pattern at this time is shown in Fig. 5. The interference pattern continued its development adding three more exterior rings and four interior rings. At the same time irregular blue gray patches developed in the broad yellow ring, which was the final transform of the single initial green ring. The final appearance is shown in Fig. 6. When

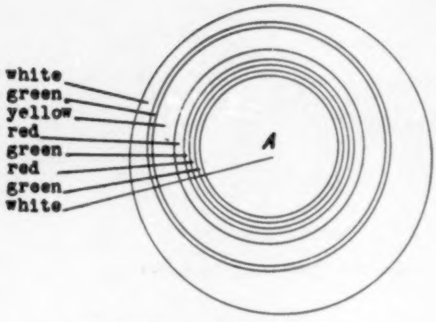

Fig. 5—Subsequent interference rings.

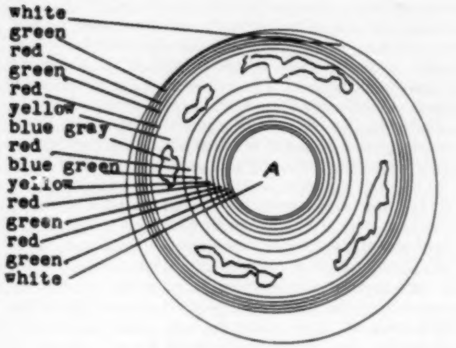

Fig. 6—Island appearance.

monochromatic green light is used, the rings form a succession of alternately black and green rings. The central portion *A* becomes black and fainter, narrower interior rings become visible. Further compression of the bubble had little effect on the interior rings or on the inner boundary of the yellow ring; the outer boundary and the exterior rings were displaced and could be made to vanish from the field of view. Decompression caused the outer boundary of the yellow ring and the exterior rings to reappear and move toward *A;* the interior rings remaining stationary until the outer boundary of the yellow ring and the exterior rings closed in upon them, then all moved in toward A.

It was most difficult to keep the galena particle clean in this experimental procedure. In general, after the pattern shown in Fig. 6 had persisted for some 5 to 10 min, an irregular white spot suddenly appeared within the yellow band and enlarged very rapidly in all directions without regard for the contour of the yellow band, leaving the field free of color and showing the characteristic appearance of galena. This white spot had disposed about it extremely narrow interference bands of similar contour. With the clearing of the field by the enlarging white spot, contact always developed.

It was virtually impossible to obtain the sequence of events recited above using galena conditioned in 30 mg per liter xanthate solution. The operator was fortunate if he could bring the galena surface into focus in time to see the flash of the enlarging white spot. With galena conditioned in very weak xanthate solutions, a portion of the sequence of events was observed.

The circumstances of this experiment are a modification of those requisite for the observation of Newton's interference rings. Two observations are outstanding, the white central spot and the spacing of the bands. In the normal Newtonian pattern the central spot is dark because a relative phase change of π occurs between the rays reflected from the two interfaces.[22] Here both reflections are of the rare-to-dense type since water has a refractive index intermediate to that of air and galena, hence no phase change takes place. In the normal Newtonian pattern, the radii of the rings increases as the root of n or as the root of $(n+\frac{1}{2})$ as n assumes successive integral values. In the pattern observed in these experiments, this orderly progression of radii is absent, in fact, the fundamental role is played by the broad yellow ring (the original green ring). There is an obvious difference in geometry of the top interface between the Newton experiment and the present one. In the former, the interface is a part of a sphere, in the latter, it is unknown. Attempts to deduce the geometry from the spacing and width of the rings have led to inconsistencies and must await the development of additional information, particularly since polarization was observed in the image. What these observations demonstrate is that the formation of the gas-liquid interface is rapid when it does occur. This supports the mechanism set forth above.

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Progeny in Comminution

by A. M. Gaudin, H. R. Spedden, and Douglas F. Kaufman

MANY studies of comminution have been made to ascertain the size distribution of the product and to evaluate the work of comminution in the light of the size distributions of the feed and product. Up to now, these studies have been essentially statistical in character, that is, a certain lot of feed was subjected to comminution in some specified way, and the aggregate product was fractionated into sizes, thereby losing all knowledge of individual relationship of feed to product pieces.

Radioactive tracers offer a means to do something in this respect which could not be done before, namely, to follow the rupturing of some particular piece in its normal environment of other pieces. That is, it permits going beyond the usual statistical limitations of size distribution studies to what may be termed a personalized or individualized study. The purpose of this paper is to present some preliminary experiments conducted with this tool.

The method employed was to mark radioactively some constituent of a feed. It is possible, of course, to consider the preparation of two lots of material of which one is radioactive and the other is not, and to blend the two ahead of the comminuting step; but to do so is open to the objection that the two preparations may not be identical. Therefore a technique has been chosen that removes this objection by merely taking out a size fraction of a comminution feed, rendering that fraction radioactive by exposure to a neutron flux, and then by returning it to the remainder of the charge for the comminution experiment.

Table I. Size Distribution of Offspring Albite Particles Originally 28/35 Mesh and in Admixture with Other Sizes After Grinding 2 min in a Steel Ball Mill

Size Fraction of Product, Mesh	Specific Activity Corrected for Background, cpm/gm (A)[a]	Weight, g (W)	Distribution in Product, Pct[b] (P)	Cumulative Distribution, Pct (ΣP)
+28	0	56.0	0	100.1
28/35	62.6	54.0	24.8	75.3
35/48	62.8	59.4	27.7	47.6
48/65	41.1	53.0	16.2	31.4
65/100	29.6	45.7	10.2	21.2
100/150	23.7	37.0	6.6	14.6
150/200	23.3	25.1	4.4	10.2
200/270	20.1	19.0	2.9	7.3
270/400	17.8	21.2	2.9	4.4
−400	22.9	25.2	4.4	—
			100.1	

[a] These activity determinations were made in rapid succession in the order given. The specific activity (A_o) of the active 28/35 mesh fraction of the feed was measured at the beginning, after the measurement on the 65/100 mesh size fraction of the product, and at the end. The decay-corrected activities at those times were 246.7, 241.0, and 236.9 cpm per gm. The weight (W_o) of the active 28/35 mesh fraction in the feed was 55.0.

[b] Example of calculation for P in the 65/100 mesh product fraction: $A = 29.6$, $W = 45.7$, $A_o = 242.7$, $W_o = 55.0$: $P = \frac{A}{A_o} \times \frac{W}{W_o} = 0.102 = 10.2$ pct.

A relatively simple procedure was developed by which albite, containing sodium, was activated in the M.I.T. cyclotron. The cyclotron makes high-speed deuterons which impinge on a beryllium target, thereby producing a concentrated neutron flux. The mineral was exposed to this flux for 2 hr. This treatment changed enough of the sodium to sodium 24 (14.8 hr half-life, 1.4 mev β) as to make detection and measurement easy. The nuclear reactions taking place were:

$$_{11}Na^{23}\ (n, \gamma)\ _{11}Na^{24}\ \text{(irradiation)}$$

$$_{11}Na^{24} \xrightarrow{\beta, \gamma, \gamma} {}_{12}Mg^{24}\ \text{(decay)}$$

The detailed technique of the experimentation was as follows: 40 kg of hand-sorted, lump albite were crushed to pass 10 mesh. After careful mixing of the lot, a screen analysis was made. The whole lot of material was fractionated on standard Tyler screens from 14 down to 200 mesh. Samples for experiments were compounded from these fractions in accordance with the screen analysis. When it was desired to make an experiment in which, for example, the 28/35 mesh size fraction was to be studied, the blend of size fractions was made as indicated above, except that the 28/35 mesh size fraction was added only after irradiation in the cyclotron.

The blended charge containing the activated albite was ground for 2 min in a laboratory ball mill with a steel ball charge of controlled size distribution. The ground product was carefully sized on a set of Tyler screens in a Ro-tap. Each size was analyzed for radioactivity by the use of an end-window Geiger-Mueller counter and standard scaling circuit. This analysis was carried out in detail as follows: a 20-g sample was placed in a Petri dish, packed carefully to obtain reproducible geometric distribution with reference to the Geiger-Mueller tube, and the activity was counted for a 2-min period.

Several determinations of the activity of the active size fraction in the feed were made at various times to establish the decay in activity with time. Linear interpolation was used to evaluate the activity that the active size fraction in the feed would have had at any given instant. The ratio of the observed activity in a size fraction of the product to the activity that the active size fraction in the feed would have had at the same time gives the fraction in the product size that came from the irradiated size in the feed.

The general formula for finding the distribution, P, of a specific individual size fraction in the feed

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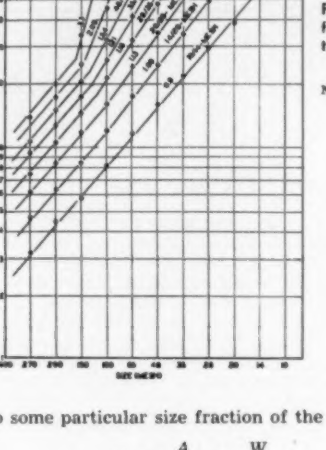

Fig. 1—Progeny of feed size fractions from 10/14 down to 100/150 mesh after 2 min of grinding.

Numbers on curves represent slopes.

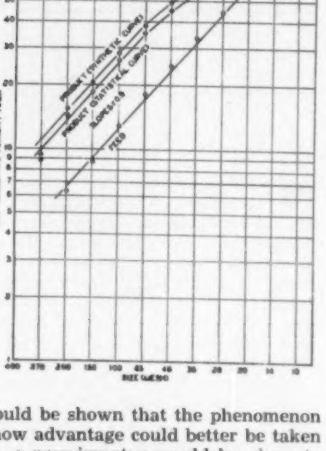

Fig. 3—Size distribution of feed and of product after 2 min of grinding.

into some particular size fraction of the product is:

$$P = \frac{A}{A_o} \times \frac{W}{W_o}$$

In this formula A and W are the specific activity and weight of the size fraction of product on which interest is momentarily focused, and A_o and W_o are the corresponding decay-corrected specific activity and weight for the active size fraction of the feed. As a typical example, reference may be made to Table I, which gives the results for one particular experiment in which the 28/35 mesh fraction was irradiated.

Study of grinding of the same feed, but with attention focused on parent particles of different sizes was made in a similar fashion. The data for all the experiments are presented in Fig. 1. This figure shows the progeny of particles of all sizes from 10 to 14 mesh down to 100 to 150 mesh. The line for the coarsest feed size (10 to 14 mesh) is like the lines obtained when a feed of one size is crushed or ground. But the lines for the finer feed sizes (say 65 to 100 mesh) suggest a behavior not recognized until now. Specifically, it suggests that fracturing one size in a normal environment of all sizes may produce relatively fewer fines than is indicated by a slope of one in Fig. 1. This is what is wanted in practice. If it could be shown that the phenomenon is general and how advantage could better be taken of it in practice, a new impetus would be given to utilitarian comminution studies. The experiments deserve checking before indulging in speculation.

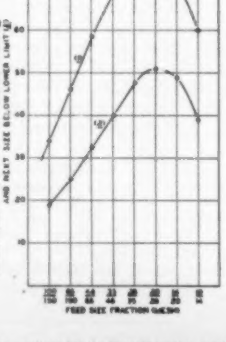

Fig. 2—Extent of size reduction for several feed size fractions after 2 min of grinding.

It is also of interest to observe that the proportion of the initially marked material that is reduced by more than a factor of $\sqrt{2}$ or a factor of 2, or any factor selected, is not constant with size but depends characteristically on it. This observation is brought out by Fig. 2. To what extent this is caused by crushing action and to the harmony between the size distributions of the feed and the grinding device selected are matters which may only be guessed at, at present.

The overall size distributions of the feed and product are found in Fig. 3. The synthetic product curve was obtained by summing up the size distributions of each size fraction after grinding and converting these results to cumulative percent finer. It is to be noted that the synthetic curve follows the actual curve as to shape but is slightly displaced. The displacement was possibly caused by the experimental error introduced by stopping the screen analysis at 400 mesh. In spite of this variation, the agreement between the curves as to slope and shape indicates that the experimental technique is valid.

The method of radioactive marking as a crushing research tool is both simple and effective. It seems that the method can be extended considerably. For example, an unusual constituent may be introduced in radioactive form in mixed feeds. Many of the variables that have been baffling in comminution operation such as comparison between crushers, pulp density, speed of mill, preferential grinding of a given mineral, etc., might also be attacked with this new tool. Finally, the method seems to offer great promise in dealing with problems involving circulating loads where there is really no suitable commonplace technique now available.

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Jaw Crusher Capacities, Blake and Single-Toggle Or Overhead Eccentric Types

by D. H. Gieskieng

THE advent of curved jaw crusher wearing plates made an approach other than segmental layout analysis desirable for prediction of capacities. For some time it had been known that the drawing board capacities of crushers using these plates had to be considerably modified by complicated experience factors to achieve agreement with results. Because these apparent capacities could be readily increased severalfold by minor crushing chamber shape changes, it was necessary that the utmost precaution be taken in predicting capacities of jaw plates modified for nonchoking, special wear characteristics, or any other reason.

To this end the laboratory and field tests outlined by the author in a previous paper[1] were made on Blake-type jaw crushers. The results of these tests were summarized in a simple first degree equation applicable to crushers using either straight or curved jaw plates. This equation first outlines the maximum capacity potential of a given crusher, then reduces this figure in accordance with installation circumstances by means of a realization factor.

It was found subsequently that this equation, with the addition of an eccentric throw factor, is applicable to standard types of single-toggle or overhead eccentric jaw crushers as far as maximum capacity potential is concerned. However, these crushers have realization factor curves somewhat different from those outlined for the Blake type.

While this paper is concerned principally with standard type single-toggle crusher capacities, the evaluation of data obtained with these machines is simplified by comparative reduction to the 10 x 7 in. Blake-type equivalents upon which the summary of the preceding paper was made. Convertibility of data from one type of crusher to the other also tends towards confirmation of both. The agreement of these data is sufficient to be considered complimentary. Consequently the feed factors, f, previously reported for Blake crushers are slightly adjusted to an average with the single-toggle crusher results.

Blake-type equation:

$$C = f \cdot d \cdot w \cdot y \cdot t \cdot n \cdot a \cdot r \qquad [1]$$

Single-toggle type equation:

$$C = f \cdot d \cdot w \cdot y \cdot t \cdot n \cdot a \cdot e \cdot r \qquad [2]$$

where C is the capacity in short tons per hour through the crusher, f is a feed factor, dependent upon the presence of fines in the feed, and the surface character of the jaw plates used.

Values of f:

	Smooth Plates	Corrugated Plates
With normal fines	0.0000414	0.0000319
Fines scalped out	0.0000368	0.0000252
Large pieces only	0.0000312	0.0000215

d is the apparent density of the broken product in pounds per cubic foot. (If the true specific gravity of the feed is known, 40 pct voids may be assumed and d becomes 37.4 times sp gr).

w is the width of crushing chamber in inches.

y is the openside setting of the crusher, in inches. In the case of corrugated jaw plates it is measured from the tip of one corrugation to the bottom of the valley opposite.

t is the length of jaw stroke in inches at the bottom of the crushing chamber. It is the difference between open and close-side settings.

n is rpm, or crushing cycles per minute.

a is the nip-angle factor. It is unity for 26° and 3 pct greater for each less nip-angle degree. A nip-angle of 20° has an a value of 1.18, and an angle of 30° has an a value of 0.88, see Fig. 1.

r is the realization factor. It is unity for perfectly uniform choke feeding and usually less for actual operating conditions according to the method of feeding used and the probabilities of hang-ups involving the size of feed and crusher opening. Approximate values are given by the curves in Fig. 2. These values are further reduced by intermittent feeding.

e is the throw or diameter of gyration of the single-toggle crusher eccentric in inches.

As evident in Fig. 1A, variation of feed size will generally have little effect on nip-angle if both jaw plates have flat areas.

Jaw plates having continuous curvature, as in Fig. 1B will have different nip-angles, depending upon the size of feed. For test work as described in this paper this effect was accounted. For general compilation of capacities for average feeds it is suggested that the nip-angle be taken at the various settings computed, at an arbitrary level, such as is indicated in Fig. 1C.

Data Evaluation

To bring the Blake and single-toggle type crusher capacity test results to common terms for evaluation, all data are converted to terms of 10 x 7 in. Blake-type performance at conditions of 100 lb per cu ft, 10 in. chamber width, 250 rpm, 0.65 in. stroke, 3-in. openside setting, and 18° nip-angle. (The nip-angle of the 10 x 7 in. Blake is 18° at 3-in. setting.) The single-toggle crusher performances are also divided by the eccentric throw to bring this effect to unity.

As outlined,[1] laboratory and field tests made on Blake-type crushers ranging from 10 x 7 in. to 60 x 48 in. were summarized along the foregoing conditions of speed, stroke, etc. This resulted in groups of data which correspond to feeds with fines, feeds

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with fines scalped, and feeds consisting of large pieces only. The results of this Blake-type summary are averaged with the single-toggle type data as converted to Blake-type equivalents, and this average forms the basis for calculation of the revised feed factors, f, i.e., considering corrugated jaw plates and feeds with fines.

$$f = \frac{19.3}{100 \times 10 \times 3.0 \times 0.65 \times 250 \times 1.24} = 0.0000319$$

where the numerator is the 10 x 7 in. Blake-type equivalent tonnage and the denominator corresponds to density, width, setting, stroke, speed, and nip-angle at the conditions which produced this tonnage.

Crushability-Capacity Effect

In the Blake-type crusher tests,[1] no capacity variation was noted for materials of different crushabilities, even though a wide range of materials was tested. These feeds had impact strengths ranging from 2.8 to 31 ft lb per in. of thickness as measured by the Bond method,[2] (potash, coke, soft hematite, limestones, traprock, taconites.)

The single-toggle crusher tests upon which this present paper is based indicate a trend in crushability-capacity effect. From Table I, which gives single-toggle to Blake-type equivalents, it is evident that feed *A*, a relatively soft gravel, resulted in capacities about 10 pct higher than those obtained with considerably tougher feeds *B* and *C*. A few tests not listed were run with very friable dry bituminous coal, which further indicated a crushability-capacity trend for single-toggle type crushers.

The simplicity of the capacity equations is maintained without loss of practical accuracy by averaging the single-toggle crusher results obtained with tough feeds (*B*, *C*, and *D*), and average feeds (*A*). It is evident that very little error is introduced for most feeds by doing this (5 pct or less). If softer than average feeds are contemplated for single-toggle crushers, up to about 10 pct additional capacity might be expected.

It is believed that the single-toggle crusher crushability-capacity trend is largely caused by the eccentric action which results in a rubbing motion between the jaw plates (attrition). With tough feeds

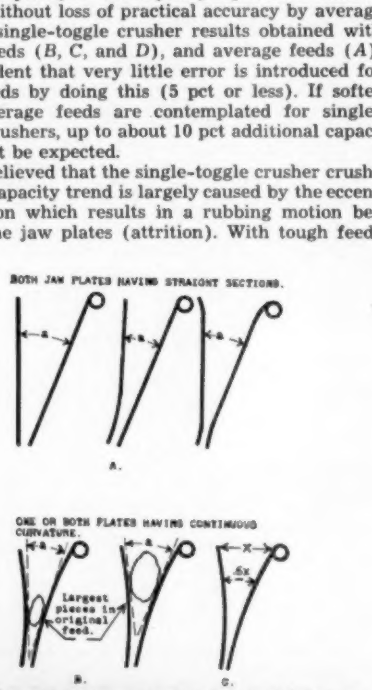

Fig. 1—Location of nip-angle measurement in Blake-type or single-toggle type jaw crushers.

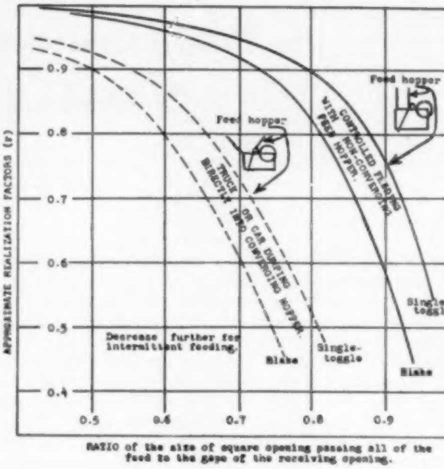

Fig. 2—Realization factors (r).

this effect is apparently negligible as far as capacity is concerned. Feed factors computed from the traprock and taconite tests came within about 3 pct of feed factors previously reported for general feeds with Blake-type crushers.

Greater differences in crushability-capacity effect than those just discussed for single-toggle type crushers have been reported by investigators working with small Dodge-type crushers. However, these crushers have rubbing motion between the jaws at the discharge, Fig. 3, and in addition have very little jaw stroke at the discharge. The crushing done by attrition between the jaws thereby assumes increasing importance with more friable feeds, as there is longer retention time in the crushing chamber caused by the small stroke and resulting discharge capacity deficiency. Comparing various data reported for Dodge-type crushers, it was found that the so-called crushability-capacity effect was very much greater with a jaw stroke of 0.04 in. than it was with 0.20 in. A further contributing effect to this tendency is believed to be illustrated in Fig. 4 of the preceding paper[1] which indicates that discharge capacity falls off more than proportionately for jaw strokes less than about ¼ in.

Reduction of data from one type or size of jaw crusher into the equivalent performance of another crusher has been accomplished by ratios of all of the various equation factors involved. The individual

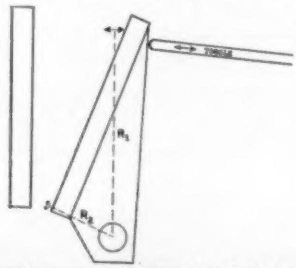

Fig. 3—Dodge-type jaw crusher showing relative swing jaw motion at feed and discharge ends.

Table I. Single-toggle Data Conversion to Blake-type Comparison Equivalents

Single-Toggle Crusher Size and Eccentric Throw, In.	Open and Closed Settings	Nip-angle	10x7 in. Blake-type comparison equivalents. 100 lb, 250 rpm, .65 in. stroke, 3 in. openside setting, 18° nip-angle, 10 in. width	
CORRUGATED JAW PLATES				
Feeds with Fines				
4¾ x 3½				
½ ecc.	0.932/0.616	16.7°	19.8 A	18.8 B (2)[a]
	0.563/0.250	19.2°	21.0 A (2)	19.5 B (2)
6⅝ x 5⅝				
¾ ecc.	1.385/0.817	19.4°	21.2 A	20.4 B
	1.037/0.470	20.9°	21.9 A	18.5 B
			20.1 Avg	
			18.5 Prev. Blake	
			19.3 Avg	
			Feed factor 0.0000219	
Feeds with Fines Scalped				
4¾ x 3½	0.938/0.622	16.6°	16.6 A	16.3 B
½ ecc.	0.563/0.250	19.2°	14.6 A (2)	13.9 B (2)
6⅝ x 8⅝	1.250/0.676	20.0°	16.1 A	13.2 B
¾ ecc.	1.000/0.435	20.9°	16.7 A	14.1 B
	0.846/0.278	21.7°	13.6 A	13.1 B (2)
36 x 25				
1¼ ecc.	3.91/3.04	26.0°		15.4 C
			15.0 Avg	
			15.4 Prev. Blake	
			15.2 Avg	
			Feed factor 0.0000252	
Feeds Consisting of Large Pieces Only				
6⅝ x 3½				
¾ ecc.	1.715/1.147	17.9°	13.9 A (3)	13.0 B (2)
			13.5 Avg	
			12.5 Prev. Blake	
			13.0 Avg	
			Feed factor 0.0000215	
SMOOTH JAW PLATES				
Feeds with Fines (−2.25 In. Slot Size)				
24 x 10				
¾ ecc.	1.64/1.10	7.5°	24.4 D	
	1.19/0.65	12.0°	24.0 D	
	0.79/0.25	12.5°	22.8 D	
			23.7 Avg	
			1.2[b]	
			24.9 Avg	
			25.0 Prev. Blake	
			Feed factor 0.0000414	
Feeds with Fines Scalped (−3 In. Slot Size)				
24 x 10				
¾ ecc.	1.64/1.10	11.0°	21.9 D	
	1.19/0.65	12.5°	21.8 D	
	0.79/0.25	14.5°	20.3 D	
	Density	**Impact Strength**	21.3 Avg	
Feed A gravel	105 lb per cu ft	15.0	1.1[b]	
Feed B traprock	105 lb per cu ft	27.0	22.4 Avg	
Feed C taconite	130 lb per cu ft	20.3	22.0 Prev. Blake	
Feed D traprock	107 lb per cu ft	23.7	22.2 Avg	
			Feed factor 0.0000368	

[a] (2), (3) indicates average of two or three tests.
[b] 5 pct added to compensate for average feeds not tested.

analysis of the extent of most of these factors is outlined in the preceding paper.[1]

Direction of Flywheel Rotation

The top of the flywheels of almost all single-toggle crushers rotate towards the crushing chamber, and this rotation is considered to be normal. Capacity eqs 2 and 3 and subsequent discussion are based upon this rotation unless otherwise noted.

Tests made with reverse rotation on standard and inverted toggle single-toggle crushers indicate about 20 to 30 pct less capacity for average crushing conditions and nip-angles of about 20°. With small ratios of reduction or larger nip-angles the capacities obtained with reverse rotation approach those obtained with normal rotation.

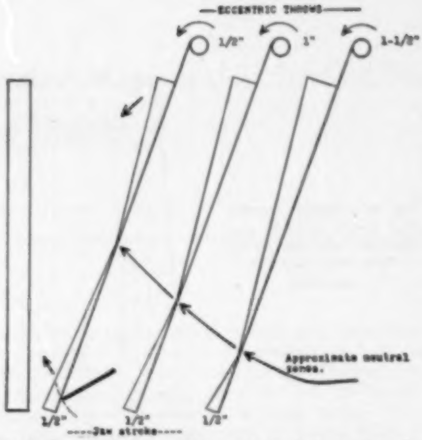

Fig. 4—Force feeding and uplifting action in single-toggle crusher.

A single-toggle crusher with a 26° nip-angle had 6 pct more capacity with normal rotation than with reverse when crushing normal feeds. However, in the particular pit where this machine was located, a large portion of stream-worn feed was present of a size corresponding to the neutral zone, Fig. 4. These boulders handicapped the crusher to such an extent that reversing the rotation and thereby dislocating the neutral zone improved the overall operation. This is believed to be an exception.

Toggle Arrangements

Standard: Almost all single-toggle crushers built today are arranged with the toggle slanting downward to the swing jaw, see Fig. 5a. This arrangement is considered standard and lends itself to strong construction as the toggle is in compression and the pressure reaction is in line with the crushing chamber.

In analyzing the jaw motion of these crushers, it may be seen that during about 50 pct of the time the opening and closing motion at the top and bottom of the crushing chamber is opposite. This results in a variable intermediate level having little crushing action, which may explain to some extent the applicability of the capacity equation to either the standard type of single-toggle or Blake-type crusher. This characteristic has some design advantage for choke feeding as the flow of material to the lower portion of the crushing chamber is limited, which tends to reduce packing tendencies in this critical region.

Inverted Toggle: Single-toggle crushers having an opposite toggle action to that of the so-called standard type are for convenience referred to as Inverted toggle crushers, Fig. 5b. This type is uncommon and the few known in the field are old.

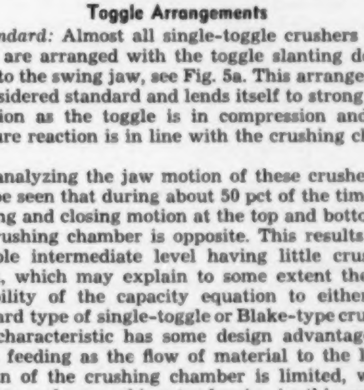

Fig. 5—Standard and inverted toggle arrangements in single-toggle crushers.

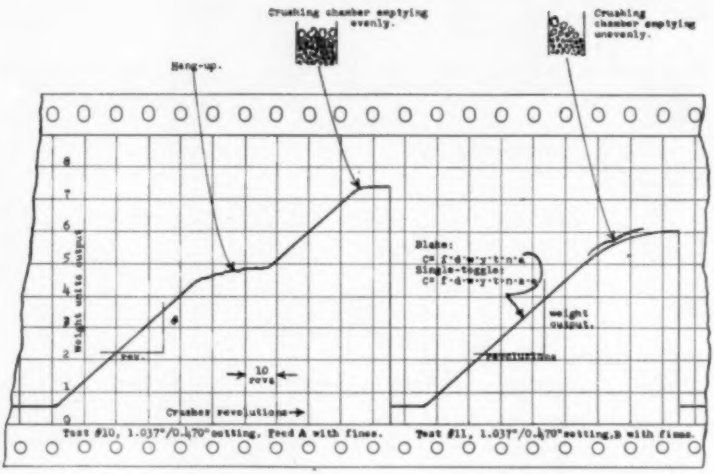

Fig. 6 — Typical crushing test records made by synchronized recording device. Power input measured separately.

An inverted arrangement may consist of either a conventional toggle in compression arranged to slope upwards to the swing jaw or of a bridle consisting of hinged tension rods on both sides of the crushing chamber. The construction of either inverted toggle arrangement is not inherently as strong as the standard type. Also, the feed to the lower chamber tends to be excessive which necessitates relatively small strokes to avoid compaction of the feed and high crusher stresses.

An inverted toggle crusher was tested to round out the single-toggle investigation. The capacity characteristics of this crusher were found to be somewhat different than the standard type of single-toggle crusher. A preliminary capacity equation based upon these tests is as follows:

$$C = f \cdot d \cdot w \cdot y \cdot (t_1 + t_2)\ n \cdot a \cdot r \qquad [3]$$

where t_1 is the stroke at the discharge, and t_2 is the stroke at the top of the crushing chamber.

Eccentricity

Single-toggle crushers with normal flywheel rotation have a crushing action caused by the eccentric motion which is commonly termed forced feeding. With standard single-toggle crushers this crowding action extends only to the previously mentioned variable intermediate level having little crushing motion. Below this level an opposite or uplifting action takes place.

The observation that a force feeding action is capacity conducive, and that an uplifting action is not, is confirmed by the results obtained in the reverse rotation experiments. It may be seen that balancing of the effects of these two actions, as illustrated in Fig. 4, is largely accomplished by the presence of the factors e and t in eq 2; as the value of e increases the proportion of uplifting action decreases.

Since standard type single-toggle crushers having eccentric throws of ½ in., ¾ in., and 1¼ in. were tested, and the results successfully converted by ratios of these eccentricities, this eccentricity factor is apparently linear for all practical purposes.

In converting standard type single-toggle data to Blake-type equivalents, the former is divided by the eccentric throw to reduce the effect described above to unity.

Realization Factor

By means of apparatus which continuously recorded the crusher tests, it was possible to eliminate the vagaries of feeding conditions as illustrated in Fig. 6, and thereby reduce the first analysis to terms of characteristic maximum capacities at given conditions of setting, stroke, etc. Various field data were then compared to the resulting equations to determine approximately what percentages of the potential capacities were obtainable in practice with various feeding methods. The effect of various feeding methods is included as a realization factor, r.

Summary

This paper is concerned with the capacity characteristics of jaw crushers, and no attempt is made to discuss the other characteristics, such as power consumption or screen analysis of product. It is assumed that in applying the equation good installation practice is followed to the extent of scalping the feed when small settings are used and employing jaw plates having a proper nonchoking curvature if feeds having unusual packing tendencies are encountered.

Acknowledgment

The writer expresses appreciation for the fine cooperation of his associates in the company and those in the field, who from the first appreciated the possibilities of the investigation and gave every assistance towards its completion. Particular acknowledgment is given Bruce Irwin, formerly of the Allis-Chalmers Basic Industries Laboratory, who supervised the tests on the 24x10 in. single-toggle crusher.

The opinions expressed on the controversial subject of crushing are those of the writer and do not necessarily coincide in all particulars with those of the Allis-Chalmers staff.

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Use of Isopachous and Related Maps in the Florida Phosphate District

by Thomas E. Wayland

AN isopachous map is one on which lines connect points of equal thickness of a given unit. This type of map is used by the Florida Phosphate Project of the U. S. Geological Survey to represent the economic phosphate deposits known as matrix and the waste material, or overburden, that overlies the matrix.

The top of the bed on which the phosphate was deposited is known as the *basement* and a subsurface contour map of this old buried erosion surface is known as a basement map. Recent experiments have been made in preparing maps that show tonnages and grades of the phosphate content of the matrix.

Few of the operating companies in the Florida phosphate district have applied isopachous (Greek *isos*, equal and *pachys*, thick) to mapping. The writer believes there is a need for the techniques discussed herein and that they can be applied to mapping other geologically similar areas in either economic or scientific investigations.

The land-pebble phosphate district of Florida occupies a compact area in the west-central part of the state. It includes mainly the following land survey divisions: Ts. 27 S. through 32 S. and Rs. 20 E. through 26 E. The town of Mulberry, Fla., is in the approximate center of the district.

The strata of the area, which is part of the Gulf Coastal Plain, occur in thin formations with broad outcrop belts, and low dips. The topography is subdued and gently rolling with three marine terraces, which are found at 30, 100, and 150 ft above sea level,[1] accounting for most of the relief. Occasional small sinkhole lakes are present, most of them above the 150-ft shoreline.

The phosphate deposits occur in unconsolidated sediments such as clays, sands, and sandy clays. They are overlain by a heterogeneous assemblage of sands, clays, muck, and iron-cemented sand, easily penetrated, in most cases, by a hand auger or drill. Limestone, locally called bedrock, or a calcareous bedclay, thought to be a residue of the limestone, directly underlies the phosphate deposits.

General Requirements

Most companies and independent prospectors operating in the district have furnished prospecting data to the U. S. Geological Survey. The information is recorded on either field logs or prospecting maps and includes the following information for each hole drilled: location of the hole, thickness of the overburden, thickness of the matrix, phosphate content in long tons per acre, grade of the phosphate content expressed as the percentage of bone phosphate of lime (P_2O_5 x 2.18) or BPL, and the percentages of iron-aluminum oxides and insolubles. The phosphate is classified according to size as either pebble or flotation material. The milling processes of the companies vary, and the size classification is necessarily different in many cases. However, pebble may be considered as larger than 14 mesh and flotation material as smaller than 14, but larger than 150 mesh. Some prospecting data include the exact depth at which bedrock or bedclay was reached, and these figures greatly increase the reliability of the data both for isopachous mapping and for mapping the basement.

A drilling density of four holes per 40 acres of land furnishes a minimum amount of data for isopachous and related mapping. From the minimum of four, densities up to 32 holes per 40 acres are used. The various drilling densities may influence the choice of the proper scale.

Selection of the proper scale is dependent upon the known drilling densities, the subsurface variations to be shown, the extent of the area to be mapped, and the detail desired in the completed map. Scales of 1:24,000, 1:4800, and 1:2400 are used in isopachous and related mapping by the Florida Phosphate Project.

The 1:24,000 scale is used most effectively with drilling densities not exceeding eight holes per 40 acres. The subsurface variations should be relatively low and uniform, permitting the use of smaller intervals without undue crowding of the lines. Comparatively large areas can be mapped on this scale, but minute detail is necessarily sacrificed, because the information is drawn from a maximum drilling density of only eight holes per 40 acres.

Isopachous and related maps of the 1:4800 scale are made of areas on which the drilling information covers from 4 to 16 holes per 40 acres. Moderate subsurface variations with relatively sharp gradations can be shown accurately. The area represented by the maps is reduced considerably in favor of detail.

The 1:2400 scale is most frequently used by the Florida Phosphate Project. It lends itself particularly well to isopachous and related mapping, being easily adapted to the multiform drilling data available. Maps of this scale are prepared with information ranging from 4 to 32 holes per 40 acres; however, use of the minimum drilling density on the

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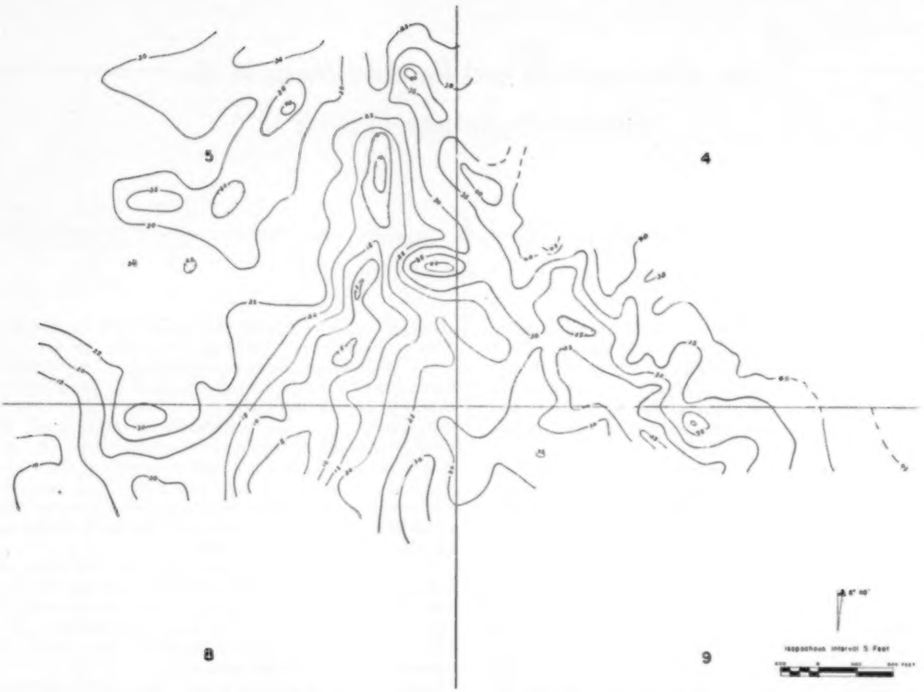

Fig. 1—Overburden isopachous map, Old Colony Mine, Polk County, Fla.

Sections 4, 5, 8, 9, T. 32 S., R. 26 E., American Cyanamid Co. The term *overburden* includes all the material overlying the phosphate deposits as well as the upper, uneconomic part of the deposit. Data from company prospecting, 1935.[2]

1:2400 scale is restricted to areas that bound or enclose adjacent areas having drilling densities of 8, 16, or more holes per 40 acres. Maps of this scale that are prepared from a minimum amount of information should be regarded as tentative, pending the availability of more complete data; on the other hand, these maps are often necessary for completeness and may show at least an outline of a pattern or trend of the strata or deposit if such should be present. Sharp gradations in the character of the subsurface are mapped conveniently on this relatively large scale. The area represented by the map is sacrificed further to show detail.

Equal thicknesses, equal altitudes, and equal tonnages are represented by lines drawn on the isopachous, basement, and tonnage and grade maps, respectively. The relatively large scales, usually uniform thicknesses of strata, and sufficient drilling densities favor a 5-ft interval in most isopachous and basement maps; however, if more detail is desired, a 1-ft interval may be used, depending upon the subsurface variations, drilling densities, and scale.

The subdivision of land in the land-pebble phosphate district of Florida is based on the Congressional Land Survey System of townships, ranges, and sections. The isopachous and related maps of the district have been drawn on plats and base maps with the section as the primary division.

A grid system of east-west, north-south coordinates, used locally by the phosphate industry, has been adopted for use in isopachous and related mapping of the Florida Phosphate Project. Sections are divided into 2½-acre squares by the coordinates, using the numbers 1 through 16 from south to north and the letters A through P from west to east. Drill holes are located usually in the center of the 2½-acre squares, although there are occasional half-line holes, or holes that fall on the lines of the grid pattern. The half-line holes are used mainly for drilling an excess of 16 holes per 40 acres.

Isopachous Maps

Maps representing equal thicknesses of both the overburden and matrix have identical modes of construction. A separate map is prepared for each area on which prospecting data are available.

Overburden and Matrix: Thickness figures are plotted in their respective positions on the base maps or plats. Lines are drawn to connect the points of equal thickness, the spaces between the lines being determined by interpolation, see Figs. 1 and 2. The isopachous maps of the overburden and matrix[2] show at a glance both the depths to which the dragline operator should remove the material and the approximate areas in which he should operate. Consistent with the nature of all maps, these may not be accurate in minute detail, but they can be of aid in determining the approximate points at which changes can be expected. Superimposition of the overburden map on the map of the matrix may reveal a correspondence of either relatively thick or

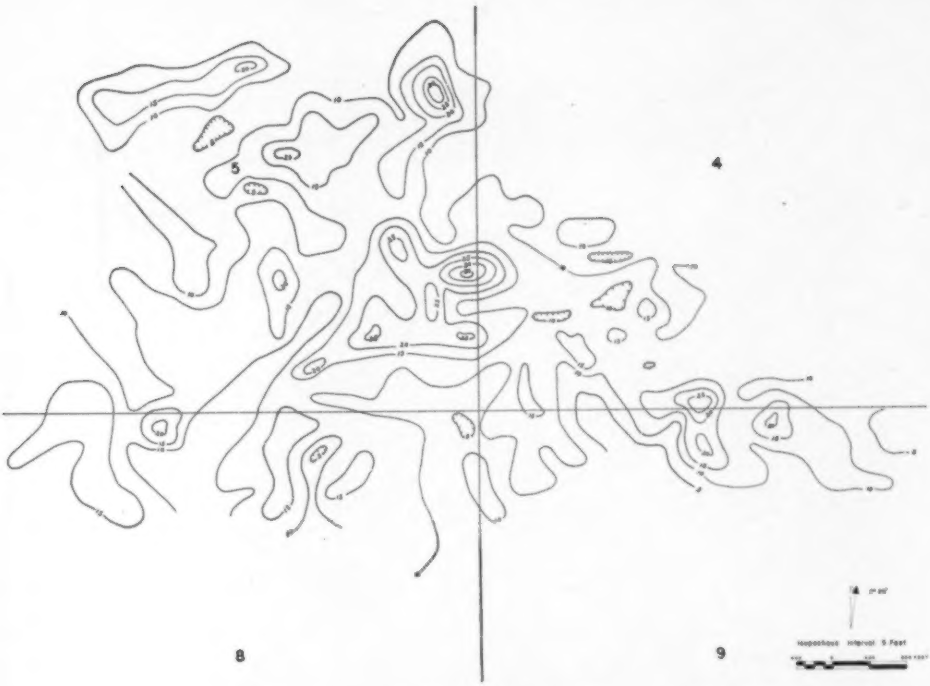

Fig. 2—Matrix isopachous map, Old Colony Mine, Polk County, Fla.

Sections 4, 5, 8, 9, T. 32 S., R. 26 E., American Cyanamid Co. *Matrix* is the term used locally to designate the economic part of the deposit. Data from company prospecting, 1935.[3]

thin areas of overburden and matrix, thereby aiding the mining engineer in the location of minable areas and possibly indicating trends of the deposit applicable to future prospecting.

Some trends of scientific interest have been noted by the members of the Florida Phosphate Project engaged in isopachous mapping. These trends are suggestive of old buried stream channels and sinks. Similar maps prepared from drilling information predicated on geologic rather than economic requirements would aid in determining the approximate thickness of subsurface formations and in plotting their lateral extent.

Basement Maps

Known points of altitude on the surface immediately over the area to be mapped are required for construction of the basement[1,2] map. Prospecting data that include the exact depth at which bedrock or bedclay was reached are the more desirable. If these figures are not given, it must be assumed that the prospector did not include bedclay thicknesses with those of the matrix.

The total depths of the drill holes (sum of the overburden and matrix thicknesses) are subtracted from the surface elevations. The resultant figures are the altitude of points at the top of the bedrock or bottom of the matrix. Subsequent plotting of these figures on a base map or plat, and the connection of points of equal altitude by contour lines, complete the basement map, see Fig. 3.

The mining engineer engaged in strip mining may want to know the altitude of the points at the top of the bedrock. A basement map can aid him materially in foreseeing some of the difficulties of drainage during strip mining and in planning his mining operations accordingly.

Tonnage and Grade Maps

The maps of tonnage and grade are prepared from the analytical results included in the prospecting data. One map represents both tonnage and grade.

Tonnage figures are plotted on a base map; lines are drawn to connect the points of equal tonnage, the spaces between the lines being determined by interpolation. An interval is chosen that represents most accurately the tonnages of the particular area being mapped.

Grade figures are now placed in their respective positions along with the tonnage figures. Again, the interval is chosen to give the most accurate representation of the grades, and lines are drawn to connect the points of equal grade. A linear pattern is used to show the areas that fall within the various grade intervals on the map illustrated in this paper, see Fig. 4. The linear pattern was chosen for publication purposes; however, a system of color shading is recommended for practical usage.

The expected recovery of phosphate by tons per acre and by grade can be estimated from a tonnage and grade map. A polar planimeter is used to compute the tonnage and grade of the areas outlined on the map by tonnage lines and grade patterns.

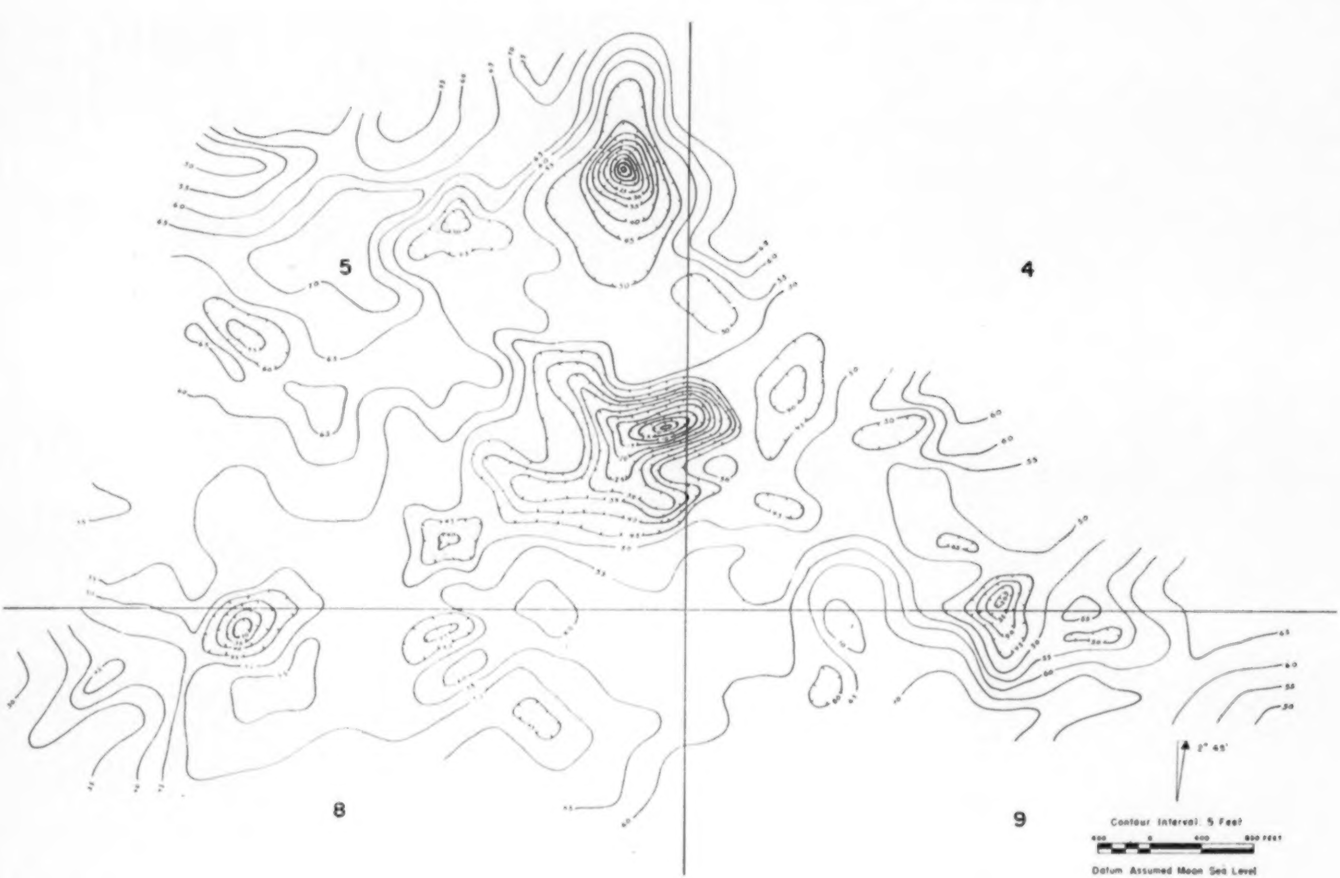

Fig. 3—Basement map, Old Colony Mine, Polk County, Fla.

Sections 4, 5, 8, 9, T. 32 S., R. 26 E., American Cyanamid Co. Contours are on the contact between the matrix and the underlying formation or bed rock. Data from company prospecting, 1935.[2]

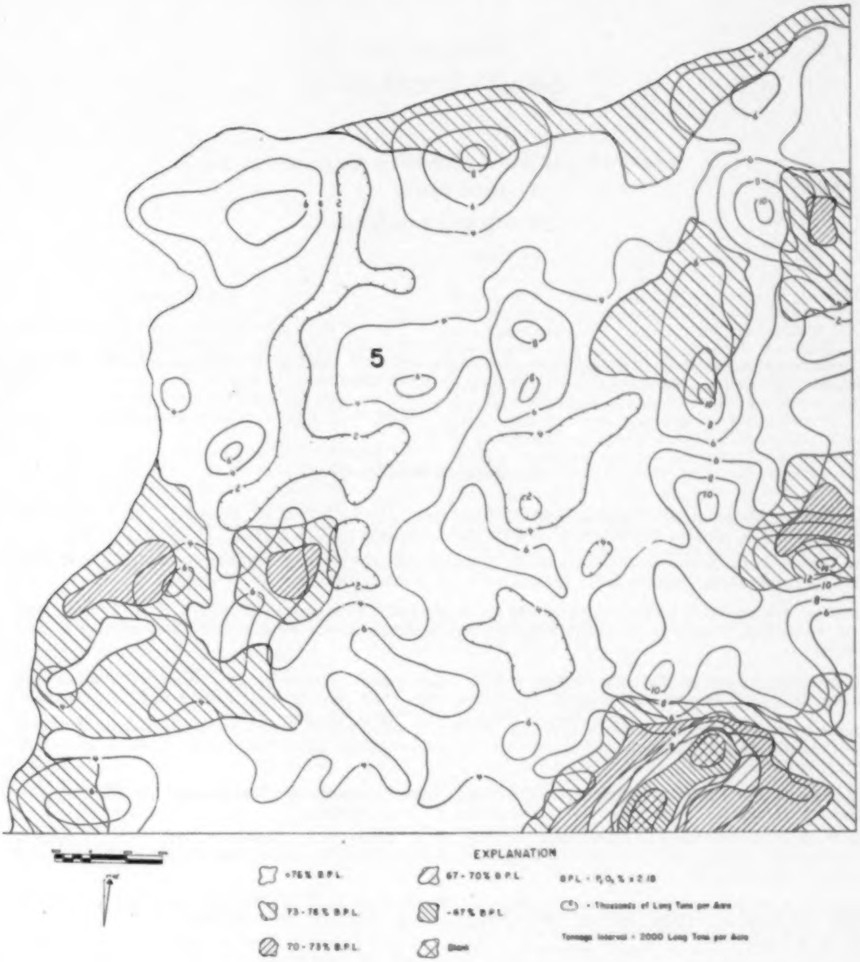

Fig. 4—Tonnage and grade map, Old Colony Mine, Polk County, Fla. Section 5, T. 32 S., R. 26 E., American Cyanamid Co.

Data from analytical results of company prospecting, 1935. Grade interval: grades are divided into 3 pct intervals from less than 67 pct to more than 76 pct BPL

The tonnage and grade map shows clearly the concentration of tonnage and grade for a given area. A tonnage and grade map used in conjunction with the overburden, matrix, and basement maps of the same area may enable the mining engineer to select the best mining location.

Acknowledgments

Owing to the serious illness of Mrs. Maher, Mr. Stuart W. Maher, a coworker of the writer, was unable to participate in writing this paper as was planned originally. The writer wishes to express his regrets of this misfortune and his appreciation of the counsel given by Mr. Maher when the paper was in an early stage of preparation.

J. B. Cathcart, Project Chief of the Florida Phosphate Project, U. S. Geological Survey, supplied the overburden, matrix, and basement maps illustrating this paper and gave valuable criticisms of its content. Arthur Crago and J. L. Weaver of the American Cyanamid Co. made the data on the Old Colony Mine available for publication. The writer would like to acknowledge also the assistance of F. N. Houser, S. L. Houser, H. L. Jicha, Louis Pavlides, R. G. Petersen, and R. H. Stewart, members of the Florida Phosphate Project.

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Discussion*

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* TP 3193

A — Metal Mining

Rock Hardness as a Factor in Drilling Problems

by W. B. Mather

DISCUSSION

R. G. Wuerker *(University of Illinois, Urbana)*—Mr. Mather is to be congratulated for stressing the most urgent need for a program of testing the physical properties of rocks, as they are encountered by the petroleum engineers on their drilling jobs and by the miners on their excavating, roof control and mineral preparation tasks.

Another valuable source of information is the *Handbook of Physical Constants.*[illegible] It stresses the physicist's approach rather than the engineer's procedure.

The study of hardness, and rock hardness in particular, is beset with many difficulties. There is no satisfactory definition of hardness yet, and the term hardness is used to describe a number of quite different properties. Hardness tests are grouped into static and dynamic determinations. They should be expressed with reference to the mechanical action of the applied force, like indentation hardness, scratch hardness, abrasion hardness, rebound hardness. A discussion of all these various hardness tests and of the special difficulties encountered in hardness testing of such heterogeneous material as rocks is given in the U. S. Bureau of Mines' standard proposal. The more critical student may be referred to the book by D. Landau: *Hardness*, which is in my opinion the best existing treatise on this subject.

[illegible] *Handbook of Physical Constants:* Geological Soc. of Amer., Special Paper No. 36 (1942).

[illegible] D. Landau: *Hardness*, the Nitralloy Corp., New York.

B — Minerals Beneficiation

A New Surface Measurement Tool for Mineral Engineers

by F. W. Bloecher, Jr.

DISCUSSION

S. Mortsell and **J. Svensson** *(Royal Institute of Technology, Stockholm, Sweden)*—Bloecher states that the apparatus described by him should be a suitable instrument in the mineral dressing laboratories for determining the specific surface of such granular products that are generally investigated.

We want to express a somewhat different opinion, as we doubt that the gas adsorption method of surface measurement really is the most suitable method in mineral dressing laboratories. It is well known that the surface area, as measured by this method, does not coincide with the surface area calculated from the size, shape, and number of particles, i.e. the superficial surface of the particles. It can be seen easily from Mr. Bloecher's own determination on ilmenite-leucoxene concentrate that the surface area measured according to this method may be several hundred times larger than the superficial surface. It is supposed that the surface of cracks, fissures, and pores within the particles also is included in the measured surface. However, this is an assumption which scarcely can be proved and certainly has not been proved.

We believe that the mineral engineer could better use a method giving the superficial surface of the particles, as in most cases this surface only is of importance to him. Such is certainly the case when calculating the resistance of the fluid flow through granular materials and the capillary forces of such materials, problems which are of great importance in connection with dewatering, filtration, and pelletizing[illegible] etc. The same may also be the case when studying flotation problems, partly since the quantities of the introduced reagents usually are so small that they can only cover the most accessible surface of the particles, and partly since the molecules of at least the collecting reagents are so large that they cannot enter the smallest cracks and pores. As to the surface, which is of consequence in the application of Rittinger's law of crushing, one is inclined to believe, from theoretical reasons and from the work by Gross and Zimmerley, that the total surface here is the most important one. However, it seems as if in that case the superficial surface was as important as the total surface.

Other reasons making the gas adsorption method less suitable for use in the mineral dressing laboratories are that the granular materials there very often are too coarse to be measured by this method, that the experimental technique is rather difficult, and that it takes a long time to measure the specific surface of a sample. Besides, the validity of the method must still be considered questionable. It is thus well known that with different gases one usually gets different values of the specific surface of the same sample, which values, even with such an ideal material as metal foils, can differ as much as about 50 pct.[illegible] Still more serious

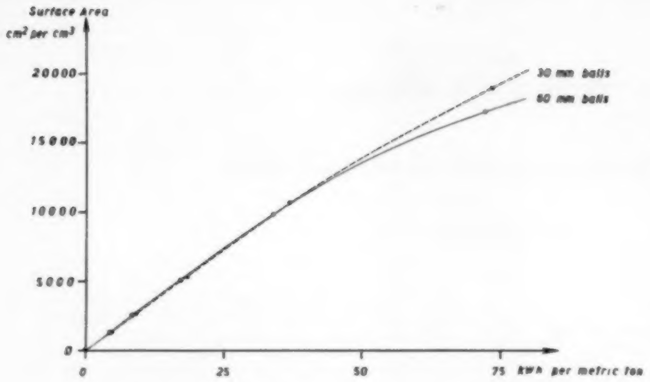

Fig. 1—Specific surface of quartz ground in ball mill as function of the net energy input to the mill.

is perhaps the fact noted by Johnson, Axelson, and Piret that the change of the temperature, from 200 to 550°C, at which their quartz samples were degassed, caused a variation of as much as 50 pct of the values of the measured specific surface.[21]

Instead of the gas adsorption method, we consider the gas permeability method better suited for use in the mineral dressing laboratory, as with that method the superficial surface can be determined. Consequently, it is possible to control and complete the surface determinations made by that method through calculations in those cases where it is possible to determine the particle size distribution of the material and the shape of the particles. For several years we have used this method in the mineral dressing laboratory of the Royal Institute of Technology in Stockholm. Before adopting the permeability method for general use, the validity of the method was thoroughly tested. During the testing it was found necessary to slightly modify the method. The modification mainly consisted in determining the permeability of the bed at several different pressures, so that both the laminar and the molecular components of the gas flow through the bed could be calculated, and in calculating the specific surface from the quotient between these two components.[22] Due to the modification, the determinations take a somewhat longer time (four determinations being made in one day by one man), but instead the specific surface measured in that way agrees closely with the superficial surface of the particles independent of the shape of these or of the porosity in the bed—at least as far as we have been able to control.

Hitherto the method has been used chiefly in connection with work on crushing and grinding conducted within the scope of Jernkontoret's (Swedish Ironmasters Association) research. We hope to publish the complete results of this work later on. However, in this connection it is interesting to note that the results indicate that Rittinger's law, although not generally valid, seems to hold true in the grinding of some homogeneous material as long as the product is not too fine. The results given in Table III from ten dry batch grinding tests will serve as a proof of this. The tests were made with balls of two different sizes on crushed quartz of the size fraction -3.7 $+2.3$ mm. The results are also plotted in Fig. 1. From the graph it is evident that the specific surface of the grinding product increases in proportion to the calculated energy input up to a specific surface of 5000-10000 cm^2/cm^3, but that the proportionality breaks down at higher values. It might be pointed out that Gross and Zimmerley in their work to verify the validity of Rittinger's law did not go to a higher specific surface than about 2500 cm^2/cm^3.

We think, however, that comparisons between surface areas measured by both the gas adsorption method and the gas permeability method would be very interesting from a theoretical point of view, especially in connection with work aiming at testing the validity of Rittinger's law. For our part, we are willing to cooperate in such a comparison by making determinations of the specific surface area with the permeability method on samples sent to us, the specific surface of which has been determined by the gas adsorption method. We are also willing to place at the author's disposal about a hundred samples, the specific surface of which we have determined by the permeability method, including samples of all the grinding products referred to in Table III.

F. W. Bloecher, Jr. (author's reply)—There are undoubtedly certain problems in which surface area measurements are most conveniently and easily made by the gas permeability method. The krypton apparatus served as a tool in dealing with problems concerning flotation fundamentals. It seems quite logical that any fissure, crack, or pore surface that is measurable by multilayer gas adsorption will also be accessible to any ions or molecules existing in water. Remember that krypton adsorption measurements described take

Table III. Results from Dry Batch Grinding Tests on Quartz with Two Different Sizes of Balls*

Size of balls, mm			60					30		
Number of balls			50					436		
Grinding time, min.	12	24	48	96	192	12	24	48	96	192
Number of revolutions	687	1375	2746	5486	10985	687	1372	2749	5480	10968
Energy input to mill motor, w-hr	49	94	191	382	793	50	101	201	404	810
Efficiency calculated	0.50	0.49	0.49	0.49	0.50	0.50	0.50	0.50	0.50	0.50
Net energy, w-hr	24	46	94	187	397	25	50	101	203	405
Net energy, kw-hr per ton	4.36	8.36	17.1	34.0	72.2	4.55	909	18.4	36.9	73.6
Percentage through 200 mesh (74.3μ)	17.4	30.3	51.5	74.0	92.4	18.5	33.5	50.8	86.7	98.8
Percentage through 400 mesh (37.5μ)	10.9	20.1	37.5	57.6	77.9	11.0	21.4	42.6	67.2	87.7
Specific surface cm^2/g	500	970	1910	3700	6510	520	1000	2020	4020	7100
Specific surface cm^2/cm^3	1330	2570	5050	9790	17230	1370	2640	5330	10630	18950

* Mill size: ∅ 305 x 305 mm
Weight of steel balls: 47.1 kg
Feed: quartz from Radanefors -3.7 $+$ 2.3 mm
Sp gr: 2.646
Specific surface: 18 cm^2/g (44 cm^2/cm^3)
Weight of feed: 5.50 kg

place at a temperature below the freezing point of the gas. Therefore, the adsorbed layer of krypton molecules is many molecules thick.

It is true that with different gases one usually gets different values for the specific surface of the material. These differences are no greater than are often noted in comparing air and water permeability measurements on the same material.[22] The observations of Johnson, Axelson and Piret concerning the effect of degassing temperature on the measured area of a quartz sample also worried me. However, time did not permit study of that problem. It would be an interesting investigation for a physical chemist.

The validity of the B.E.T. method of surface area measurement has been quite well substantiated by the work of Harkins and Jura[24] who devised a so-called absolute method of surface measurement involving no assumption concerning the area occupied by one molecule of gas. They accurately measured the heat evolved upon immersion in water of a crystalline solid that had been previously exposed to saturated water vapor so that the surface energy of the outer film on the solid was the same as that of water. The surface area of the solid is given by the ratio of the heat evolved to the surface energy of water after a small correction has been made for the thickness of the water film. Excellent agreement between the Harkins-Jura and the B.E.T. method has been noted.

Comparisons between gas adsorption and permeability surface area measurements would indeed be interesting.

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A New Theory of Comminution

by F. C. Bond and J. T. Wang

DISCUSSION

H. J. Kamack *(E. I. du Pont de Nemours & Co., Inc., Wilmington, Del.)*—Rittinger's law usually is stated to the following effect: "The work (or energy) consumed in particle size reduction is proportional to the new surface area produced." The law has been stated substantially in this way by Taggart,[20] Berry,[21] Dalla Valle,[22] Coghill and DeVaney,[23] Richards and Locke,[24] Gross,[25] and many others, and, according to Gaudin,[26] was originally expressed by Rittinger in the same form. Consequently there can be little doubt that this is the understanding of the law among most workers in the field of particle size reduction. Bond and Wang, however, express the law in the form "the *useful work accomplished* . . . ." (italics theirs). The distinction is critical, for in the form used by Bond and Wang the law becomes, as they themselves remark "merely a more or less arbitrary definition of useful work," while in its usual sense the law expresses a physical hypothesis which has been verified experimentally within certain limitations.

Considering the way the law has been used, it might be stated more explicitly as follows: "In a given machine operating under a given set of constant operating conditions, the work consumed in the particle size reduction of a given material is proportional to the new surface area produced." Or, as Coghill and deVaney have said,[27] "the (Rittinger) law holds only when the tests being compared are made under analogous conditions." There are occasions when the law transcends these limitations; for example, the surface area produced per unit energy consumption for a given material in a ball mill does not vary much over a fairly wide range of operating conditions. But by and large, the surface area production per unit energy consumption will vary with the operating conditions, the type of machine, and the material. The essence of Rittinger's law is that the surface area production per unit energy consumption is independent of the particle size, and this has been verified experimentally by numerous workers for numerous materials, within certain limits. An important limitation is that when one grinds to very small particle sizes, agglomerative forces may tend to interfere with size reduction so that the surface area increases less rapidly than Rittinger's law would predict. Within such limitations, Rittinger's law can be regarded as empirically established.

The law has, however, certain theoretical implications, and it seems to be chiefly against these that Bond and Wang direct their criticism. Solids are believed to possess surface energy which is proportional to surface area. Thus, Rittinger's law implies a proportionality between surface energy produced and mechanical energy expended (for a particular material in a particular machine). It does not imply that all or most of the mechanical energy is transformed into surface energy; in fact it is known that most of the mechanical energy is transformed into heat. Bond and Wang assert that most of this heat arises from the damping of elastic vibrations of stressed particles. This may possibly be true for crushing (with which they are chiefly concerned), although in grinding it is probable that much of the heat arises from friction between particles. However, the fact that the surface energy is small compared to the heat energy does not invalidate Rittinger's law, which implies merely that they are proportional.

The authors also criticize Rittinger's law on the grounds that "this theory cannot be justified mathematically, since work is the product of force times distance, and the distance factor is ignored," and "energy input must be the product of force times distance, and the Rittinger theory completely ignores large variations in the distance (strain dimension or deformation) throughout which a force must act to produce breakage of different materials." However, the quantities force and distance as such are irrelevant to Rittinger's law, which considers energy input as an integral quantity. The fact that different materials require different forces and strains (hence, different amounts of energy) to break them is incontrovertible but again is irrelevant to Rittinger's law, which, as mentioned before, applies only to a single material. Even in theory, it would not be expected that the surface area produced per unit energy consumption would be the same for different materials, because the surface energy per unit area is presumably specific to each material.

Bond and Wang advance another argument of a

Table VI. Grinding Test of Sand

Grinding Time, Min	Specific Surface Average Particle Diam, Microns[a]		New Surface Area, Sq M		80 Pct Size, Microns[b]	
	35/48 mesh sand	100/150 mesh sand	35/48 mesh sand	100/150 mesh sand	35/48 mesh sand	100/150 mesh sand
0	349	124	0	0	360	130
60	7.1	8.5	840	684	53	68
120	4.07	4.5	1400	1291		

[a] Measured with a Fisher Sub Sieve Sizer with results corrected for *slip flow*. Feed sizes determined from sieve analyses.
[b] From plot of sieve analysis.

different nature, saying that "The Rittinger theory . . . . has been criticized as assigning too much energy to reduction of the very fine particles, while requiring the use of a more or less arbitrary grind limit." This criticism should properly be applied to the method employed for determining surface areas; i.e., calculation from size distributions, using the Gaudin-Schuhmann law for extrapolation. It is a consequence of this law that, if the slope of the cumulative distribution curve is less than 1 (as is usually the case for comminuted materials), then the total surface area on any size grade increases without limit as the particle size of the grade decreases, so that, unless some arbitrary limiting particle size is assumed, the calculated surface area of the powder is infinite. This would indicate that the Gaudin-Schuhmann relationship is physically untenable, even though it affords a fairly good approximation to part of a size distribution. Equally good or better approximations can be obtained with other methods of plotting, notably the method of plotting on log-normal (also called log-probability) paper.[illegible] Such paper is available commercially.[illegible] A straight-line extrapolation on this paper will always yield a finite surface area for the powder, and areas so calculated usually agree fairly well with direct measurement of surface area (e.g., by the permeability method).

Strain-Energy Theory: According to this hypothesis, the work consumed in size reduction of a particular material is proportional to $(n + 2)(n - 1)/n$, where n is the reduction ratio.* This expression is approximately equal to n (the difference never exceeds 12.5 pct of n when n is greater than 1.66). When the cumulative size distributions of feed and product are nearly parallel, Bond and Wang define n as F/P, where F and P are the sizes which pass 80 pct of the feed and product, respectively. When the feed is closely sized, Bond and Wang describe a method for calculating an average value of n. In either case, it can be shown that n represents approximately the ratio D_f/D_p, where D denotes specific surface average particle size and the subscripts f and p refer to feed and product, respectively. D (in cm) is defined by the relation

$$D = 6/(SA)(Sp)$$

where SA is the specific surface area of the material in sq cm/g and Sp is its specific gravity.

According to the strain-energy theory, then, the work consumed in size reduction of a particular material is proportional to D_f/D_p, while according to Rittinger's law it is proportioned to

$$\frac{1}{D_p} - \frac{1}{D_f}$$

Consequently, if one starts with a given feed size, and comminutes until the reduction ratio is fairly large, both Rittinger's law and the strain-energy law predict that the energy consumption will be inversely proportional to the specific surface average particle size of the product. (Also, if the slope of the cumulative size distribution curve of the product is assumed to remain constant as comminution progresses, then the energy consumption will be inversely proportional to F). This is why in Fig. 2 of the paper the curves for the *modified* Rittinger method and the strain-energy method are parallel, both for *crushing* and for *grinding*.

On the other hand, if one were to start with two different feed sizes, of the same material, and apply the same amount of energy to each, sufficient to produce a fairly large reduction ratio, the Rittinger law would predict that the product sizes would be nearly equal, whereas the strain-energy theory would predict that the product sizes would be proportional to the feed sizes. This suggests a simple crucial test of the theories, which we have made as follows. Two closely-sized grades of sand, one 35 to 48 mesh, the other 100 to 150 mesh, were ground dry for the same length of time under the same conditions in a 12 in. diam x 6 in. ball mill (speed, 51 rpm; charge, 47 lb of ¾ in. steel balls, 5.9 lb of sand). The ratio of the mean feed sizes, therefore, was 2.8 to 1.

The measurements of surface area and specific surface average particle size are in approximate agreement with Rittinger's law but in violent disagreement with the strain-energy theory. Also, the 80 pct sizes of the products after 1 hr are 53 and 68 microns, again in violent disagreement with the strain-energy theory. (The fact that the coarser feed gave a slightly finer product we attribute to a small difference in grindability of the two grades.)

Evidence Adduced for the Strain-Energy Theory: Bond and Wang offer Fig. 2 of their paper as evidence of the correctness of the strain-energy theory. This graph is a log-log plot, with energy consumption per unit weight of material as ordinate and with the expression $X = F^{1/2}/P^{3/2}$ as abscissa. Empirical data yield a set of curves with a slope of 0.5. The strain-energy theory may be written

$$h = kF/P$$

where h is the energy consumption and k a constant characteristic of the material. This may be written as

$$\log h = \log k + \frac{2}{3}\log F + \frac{2}{3}\log X$$

which shows that on the graph the strain-energy theory is represented by a straight line with a slope of 2/3. This disagrees with the empirically determined slope of ½. That the strain-energy curve for crushing lies in the same region as the empirical data is fortuitous because, as the equation above shows, the actual position of the curve on the graph may be shifted at will by varying the feed size, F. This is also indicated in the figure itself, because the curves labeled "crushing" and "grinding" represent two arbitrary choices of feed size.

The curves for the Rittinger law may also be freely translated on the graph by varying the feed size. For fairly large reduction ratios, the Rittinger curves nearly parallel the empirical curves. The reason for this is that the surface areas were calculated upon the Gaudin-Schuhmann law, and it can be shown from this that D_p is approximately proportional to P^m. Thus, Rittinger's law:

$$h = k\left(\frac{1}{D_p} - \frac{1}{D_f}\right)$$

becomes, for a fairly large reduction ratio

$$h = k'/P^m$$

or

$$\log h = \log k' - \frac{1}{3} m \log F + \frac{2}{3} m \log X$$

so the slope is $\frac{2}{3}m$. Bond and Wang use a value of m

* The notation of Bond and Wang is followed throughout this discussion.

= 0.7, so the slope is 0.47, or nearly the same as the empirical slope. Thus one could conclude that the Rittinger law agrees better than the strain-energy theory with the empirical data presented by Bond and Wang.

The *modified* Rittinger law, which Bond and Wang also plot, consists in replacing D_p and D_f with P and F in Rittinger's law. Thus, for a fairly large reduction ratio, the modified Rittinger law can be expressed as

$$h = k/P$$

or

$$\log h = \log k - \frac{1}{3} \log F + \frac{2}{3} \log X$$

This curve, therefore, parallels the strain-energy curve, as has already been noted.

Conclusion

The arguments which Bond and Wang present against the Rittinger theory do not stand up under critical examination. The "strain-energy" theory which they propose instead is not supported by the empirical data they present and is disproved (for grinding) by an experiment which shows that for sand the energy required for grinding is not necessarily proportional to the reduction ratio.

R. G. Wuerker *(University of Illinois, Urbana)*—The strain-energy theory, as used by the authors, is based on the use of the modulus of resilience as a criterion for the failure of a material. The quantity of strain energy stored per unit volume, as expressed in the term $S^2a^2/2E$, step 1, has been proposed as a basis for determining failure as far back as 1885 by Beltrami.[illegible] Its application to failure of a mine roof has recently been discussed by Philipps.[illegible] Its use in the study of comminution, steps 2 and 3 in the authors' proposal, is therefore well founded and full of promise.

If the modulus of resilience should be correlative with quantities used in comminution, like impact crushing strength, energy per new surface created, etc., much time and expense could be saved. For the area under the elastic part of the ordinary stress-strain diagram is equal to the modulus of resilience and could be easily graphed or computed.

However, data listed in the literature do not allow such a correlation attempt yet. One never finds crushing and grinding data of rocks, of which also fundamental properties like modulus of elasticity and compressive strength, or vice versa are known. It would be indeed a worth-while research project to make the necessary measurements and correlations of rocks considered typical for their behavior in crushing and grinding.

The authors have stressed the need for more accurate determinations of the modulus of elasticity. Besides accuracy, an agreement is needed, which modulus we intend to use. For with brittle materials like rocks, E is not a unique quantity, not a constant, as in the case of ductile metals. The stress-strain curve has very often no straight line portion at all. In the case of concrete it is customary to work 1—with the tangent modulus through the point of origin, 2—with the tangent modulus for a given stress, and 3—with the secant modulus at the ultimate limit stress.

These three moduli are arrived at by static testing. But recently sonic testing has come to increased use. E determined by this latter method is always the initial tangent modulus through the point of origin and is therefore a higher value than many moduli of elasticity listed in the literature which have been obtained by the static method. In a recent publication of the U. S. Bureau of Mines[illegible] are some examples (group numbers 29.2 and 30.1) where E obtained by the sonic method is three or four times as high as the values obtained by the conventional static method. If one is going to use indiscriminately any modulus of elasticity, without stating how it has been obtained, not much accuracy can be expected.

It is suggested to reach an agreement first, which E should be used in the case of comminution computations.

F. C. Bond (authors' reply)—The distinction between total work input and useful work accomplished in the statement of the first form of the Rittinger theory is important, since it is obvious that in a machine whose mechanical efficiency is different at different product sizes the work consumed in particle size reduction is not proportional to the new surface area produced; unless it should happen that the true relationship between useful work accomplished and surface area produced should compensate for the difference in mechanical efficiency. In a given machine operating under a given set of constant operating conditions the absolute mechanical efficiency is not necessarily constant at different feed and product sizes.

The power required to rotate a ball or rod mill when the feed is cut off is approximately the same as that required when grinding, and it is difficult to imagine that when 99 pct of the feed is cut off the relationship between energy input and surface area production remains constant.

I seriously question the statement that "Within such limitations, Rittinger's law can be regarded as empirically established." Most of the published experimental confirmations of the Rittinger theory start with an unnatural feed of particles of constant size, and often of uniform size, from which the fines have been removed. A valid confirmation of the theory requires that a feed be used with the same normal plotted size distribution slope as the product; with a scalped feed the relative amounts of work done on the different size fractions of the product are entirely disproportionate. Tests made with a scalped feed may appear to confirm the Rittinger theory while concealing the true theory from discovery. To claim that a scalped feed makes no difference is to assume a priori the proposition to be proved. In considering confirmatory tests made on quartz it should also be remembered that grindability tests on pure crystallized quartz[illegible] do not parallel those made on most ores, the quartz being relatively harder to grind at fine mesh sizes than average materials. There is also important experimental evidence that the theory does not hold.[illegible]

The second form of the theory involves the concept of surface energy, and two schools of thought exist on this subject.

One believes that all of the useful input energy is transformed into surface energy. It is exemplified by the statement of Professor Gaudin that "—the efficiency of a comminution operation is the ratio of the surface energy produced to the kinetic energy expended." If this is true, we can hope for some wonderfully welcome decreases in the energy required to crush and grind, since present efficiencies under this definition are around 1 pct or less.

The other school believes that the surface energy produced is a constant, though probably small, proportion of the energy required. However, it has never been clearly explained just why the ratio should be constant, except that the logic of the Rittinger theory demands it. It appears much more accurate to depend upon the total energy input, rather than upon the constancy of the small proportion of it represented by the surface energy.

If the energy required to form a unit of new surface area of different materials is directly proportional to the unit surface energies of these materials, and the energy required for different size reduction is directly proportional to the new surface produced, then the Rittinger theory applies to all materials; and if the energy required to break is the product of average force times the distance of deformation it appears that the energy required cannot be independent of the amount of deformation, as the Rittinger theory requires.

The work of Carl Zappfe[illegible] supports the existence of micelles of about colloidal size in metals, and it would seem logical that they exist in rocks as well,

thus postulating the existence of a grind limit to the Gaudin-Schuhmann distribution line. It is noteworthy that the calculated total surface areas of ground products by any of the "absolute" methods are similar to those obtained by the G-S line and a grind limit of 0.7 micron, with a suitable correction for the surface areas of irregular particles.

If the G-S line is straight, the log-probability plot of the line is curved. The product of an open circuit grind product may sometimes yield a straight log-probability line and a curved G-S line, but this probably results from slabbing of the larger particles and tramp oversize. The G-S line of a closed circuit product is usually straighter than the log-probability line, and the G-S plot is more generally used.

Since the paper was written I have appreciated that some of the same objections apply to the strain-energy theory that apply to Rittinger and Kick: namely, all are theoretically explained by the regular breakage of cubes, which is not similar to actual crushing and grinding of irregular pieces of rock; and all deal primarily with increments of work from the feed size, rather than with the total work from theoretically infinite particles of feed.

In Mr. Kamack's grinding test the confirmation of the Rittinger theory was made in the usual manner previously criticized; an unnatural feed of sized particles was used. The fact that the coarser sand feed gave a finer product size and more surface area after a constant work input shows that considerable difference exists between the grindabilities of the two grades of sand. One calculation indicates that the 100/150 mesh sand is 2.66 times as hard to grind in the same size range as the 35/48. It is obvious that no valid comparison of the theories can be made between two such different materials.

Fig. 2 was prepared to illustrate objectively the comparison of the different theories, and not as evidence of the correctness of the strain-energy theory. The difference in slope between the empirical lines and the strain-energy lines, and the lateral displacement of all the theoretical lines with changes in the feed size, are all shown in the figure, and it is true that one might conclude from Fig. 2 that the Rittinger lines may agree as well with the empirical data as the strain-energy lines. We did not attempt to prove the correctness of the strain-energy theory in this paper, but presented it as fairly as possible as a logical alternative to the Rittinger and Kick Theories. It is a development of the laws of mechanics dealing with the breakage of physical structures, and as such is worthy of consideration. None of the theories agrees too well with data from actual plant operation.

The Rittinger theory superficially appears logical, but the real test of a theory lies in its practical application. After many years of intensive study and effort it has still not been correlated satisfactorily with operating results. It remains practically valueless in the solution of actual crushing and grinding problems, which must still be solved by empirical comparisons within narrow size ranges. A correct theory, which will permit direct comparisons of efficiency between different materials in different machines over all size ranges from easily obtained data, is needed. The strain-energy theory resulted from a logical attempt to fill this need, and we believe it is a step in the right direction. Development of the correct theory will permit numerical evaluation of the many diverse factors which affect crushing and grinding, and result in more accurate predictions regarding new installations, leading to the development of more efficient comminution methods. General acceptance of the Rittinger theory may have retarded discovery of the correct theory, and retained the industry in the empirical darkness in which we are all now groping. The light certainly exists, and the strain-energy theory represents one attempt to find it.

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[37] Carl A. Zappfe: Structure Interne de L'etat Solide; *Revue de Metallurgie* (1950) **47**, No. 5.

Comparative Results with Galena and Ferrosilicon at Mascot

by D. B. Grove, R. B. Brackin and J. H. Polhemus

DISCUSSION

P. L. Jones *(Sink and Float Corp., New York)*—The comparisons between galena and ferrosilicon medium should be applied only to the specific sink-float process used at Mascot since no evidence is presented that allows the comparison to be extended generally to galena medium as now used in other forms of sink-float.

It is stated that flotation recovery of galena medium was tried but was unsuccessful and therefore the elaborate gravity system of medium recovery was applied. The expense of the gravity system at Mascot was one of the reasons for changing to ferrosilicon. However two important sink-float plants treating a combined total of over 13,000 tons of ore per day are using flotation to clean galena medium most successfully and are obtaining medium recoveries comparable with those reported in the paper for ferrosilicon.

Slightly increased zinc recovery is claimed due to the change-over but it is to be noted that at the time of the change a decreased density differential was obtained in the separator. With galena a differential

Table XII. Based on Results in Table II in the Paper

Product	Weight, Pct	Zn Assay, Pct	Zn Distribution, Pct
Sink	21.6	10.33	90.5
Float	78.4	0.297	9.5
Cone Feed	100.0	2.46	100.0

Table XIII. Theoretical Results Based on Float Elimination Retaining Actual Float Elimination But Using Zinc Distribution Figures Obtained in the Tests.

Product	Weight, Pct	Zn Assay, Pct	Zn Distribution, Pct
Sink	21.6	10.64	93.3
Float	78.4	0.21	6.7
Cone Feed	100.0	2.46	100.0

density of 0.15 was used.[1] Although not so explained in the paper, this decreased differential is presumably due to the fact the new galena medium was —10 mesh whereas the new ferrosilicon medium is —65 mesh. The effect of a high density differential is to decrease mineral recovery as suggested by work done at the Central Mill of the Eagle-Picher Company at Picher, Oklahoma.[2] Marginal sink particles are apparently prevented by the high density differential from reaching the sink product outlet.

The figures in Table XII were taken from Table II and show the good metallurgical work done.

To compare with these results, heavy liquid separation tests were made on composite samples during two periods separated by eleven months and gave remarkably similar results, suggesting there is but little variation in average ore. These are shown in Fig. 5 in the paper. Using these results it is possible to compare theoretically perfect density separation with actual as given in Table XIII.

These results illustrate that however good the results seem they could be bettered. Thus if a 78.4 pct float elimination as actually obtained was acceptable, with theoretical separation a float tailing of 0.21 pct Zn should have been obtained, Table XIII; whereas actually 0.297 pct Zn was obtained, Table XII. Looked at another way, if a 90.5 pct Zn recovery as actually obtained was acceptable, with theoretical separation an 84 pct elimination of float should have been obtained, Table XIV, but actually 78.4 pct elimination was obtained, Table XII.

These results illustrate that looked at either way the sink contains considerable float and the float contains considerable sink. It seems that this can be expected in any sink-float process that uses a density differential as large as that used at Mascot even after the changeover.

D. B. Grove, R. B. Brackin, and **J. H. Polhemus** (authors' reply)—The purpose of the paper was to show the major improvements resulting from the conversion to ferrosilicon at Mascot. No attempt was made to extend the comparison between galena and ferrosilicon to other forms of sink-float. However, this could readily be done to the extent that wherever ferrosilicon could be substituted for galena, medium costs would be reduced because of the lower cost of ferrosilicon, even if no reduction in medium consumption were effected.

The flotation system of cleaning medium at Mascot was not as effective as the gravity method or it never would have been abandoned. It is a fundamental fact that galena used in a medium system is exposed to oxidation, which complicates its recovery by flotation. The ferrosilicon loss at Mascot is 0.15 lb per ton of ore milled. On the same ore the galena loss was 0.80 lb per ton milled. There can be no question as to the reliability of this figure, since the Mascot ore is lead free, making it much easier to determine the exact medium consumption than in a plant where the galena is being obtained from the ore milled. It would be of interest to see the exact figures on medium consumption from the two large plants using flotation to clean galena and obtaining medium recoveries comparable to the 99.9 pct recovery of ferrosilicon made at Mascot. Even if this recovery could be equalled with galena, the unit cost of the latter is approximately double that of ferrosilicon, which gives a much higher medium cost.

Table XIV. Alternatively Theoretical Results Based on Actual Zinc Distribution, i.e. Retaining Actual Zinc Distribution but Using Weight Distribution Figures Obtained in Tests.

Product	Weight, Pct	Zn Assay, Pct	Zn Distribution, Pct
Sink	16.0	13.94	90.5
Float	84.0	0.28	9.5
Cone Feed	100.0	2.46	100.0

Table XV. Metallurgical Results for August 1951.

	1947, with Galena	First 8 Months 1950, with FeSi	August 1951
Tailing Assay, Pct Zn	0.310	0.297	0.264
Concentrate Assay, Pct Zn	12.08	10.33	12.06
Heavy Media Separation Recovery, Pct	89.38	90.22	90.89

The expense of the gravity cleaning system at Mascot was but one reason for changing to ferrosilicon. The primary reason was the high cost of galena and the difficulties involved in handling this soft, friable medium which is inherently more difficult to recover than a magnetic medium.

The higher differential in the cone with galena was due to the necessity of using 10 mesh galena. Finer sizes comparable to the 65 mesh ferrosilicon now in use were tried but ruled out because of exorbitant losses of fine galena. Generally speaking, the finer the galena used, the higher will be the medium loss. This factor had to be balanced against any reduction in tailing assay that might have been obtained by operating with a lower differential. Three years operating experience with ferrosilicon have shown that the best results on Mascot ore are made with a differential of 0.08 to 0.10 in the cone. It is our belief that the question of whether or not differential is required in a sink-float operation and the extent of this differential can be determined only by actual operation on the individual ore, and that no generalizations on this question are possible.

The plant is currently treating +¼ in. ore, a much finer feed than the galena circuit was able to handle, and the results are the best that have been obtained in the history of the operation. The metallurgical results for August 1951 are shown in Table XV beside the comparable figures given in the paper, indicating that the further improvements in metallurgy predicted in the paper are being obtained.

It is recognized that the results being obtained could be bettered, as Mr. Jones points out. This is true in any ore dressing process. However, the above tabulation shows that marked improvement has been made since the paper was written, and this improvement is in no way due to any change in differential. Rather, it is due to minor changes in operating technique and increased use of recording instruments to keep the operators informed of tonnage rate, medium gravity and medium volume. Other refinements of this nature may bring about further improvements.

[1] Differential Density Separation at Mascot, Tennessee, Engineering and Mining Journal (July 1940).

[2] U.S. Bureau of Mines, R. I. No. 4511, May, 1949.

The Probability Theory of Wet Ball Milling and Its Application

by Elliott J. Roberts

DISCUSSION

F. C. Bond *(Allis-Chalmers Mfg. Corp., Milwaukee)*—This paper considers comminution as a first order process, with the reduction rate depending directly upon the amount of oversize material present. The data show that other factors should be taken into account, and it is possible that in time these may be evaluated as simultaneous or consecutive reactions. Development of the theory of comminution has been retarded for many years by the assumption that surface area measurements constitute the sine qua non of the work done in crushing and grinding, and it is encouraging to note the belated growth of other ideas.

In the Abstract the term "net power" should be changed to "net energy." Throughout the paper the term "hp per ton" should be changed to "hp hrs per ton", or "hp hr t."

The term "Probability Theory" in the title does not seem appropriate, since it is not clear how the probability theory is used in developing the ideas in the paper.

There seems to be a contradiction between the large calculated advantages of closed circuit operation and the statement following that the closed circuit test results showed no significant change in grinding behavior, when compared with the batch grind curves.

Tables I and II show that between 75 pct and 50 pct solids the energy input required decreases with increasing moisture content and may indicate the advisability of grinding at higher dilutions in certain cases.

The calculation of the hp-hr per ton factor indicates an input in the laboratory mill of only 7.32 gross hp per ton of balls; this casts some doubt upon the accuracy of the factor used, since the power input in commercial mills at 80 pct critical speed is customarily much higher.

The tests show that within fairly wide limits the amount of ore in the laboratory mill may be varied and a product of constant fineness obtained, provided that the grinding time is varied in the same proportion. This has often been assumed, and confirmation by actual testing is of value.

The C_{avg} corrections for differences between the plant and laboratory size distributions do not seem very satisfactory, since in many cases the plant/laboratory ratio is farther from unity after correction than before.

The following equation has been derived from the data in Table VI:

$$\text{Relative Energy Input} = \frac{(\text{log new ball diam in in.} + 0.410)}{0.844}$$

from which the relative energy inputs for balls of different sizes can be calculated and compared. The relative energy input is unity for balls of 2.715 in. diam. The equation indicates that the work accomplished by a ton of grinding balls per unit of energy input is roughly proportional to the square root of the total ball surface area; provided, of course, that the balls are sufficiently large to break the material. The data in support of this statement are admittedly meager, but are fairly consistent when plotted.

The relative grindability values listed in Table VI for 200 mesh multiplied by 4/5 apparently correspond approximately to the A-C grindability at 200 mesh.[1]

It would seem that for open circuit tests comparable accuracy could be obtained much more simply by the old method[2] of plotting the test grind, extending the mesh grinds to the left of zero time if necessary, and determining from the plot the equivalent time required to grind from the plant feed size to the plant product size, using the average of several mesh sizes. The energy input value of one time interval could be determined by tests on materials of known grinding resistance, and this multiplied by the interval required should give the desired energy input value. The relative grindabilities would be the relative time intervals required for a specified feed and product size.

When the plotted mesh size lines of a homogeneous material are extended to the left beyond zero time they meet at one point at zero pct passing. The horizontal distance of this point from zero time indicates the equivalent energy input required to prepare the mill feed.

The author's results show that the closed circuit grinding tests give about the same K values as open circuit tests, from which he concludes that open circuit tests are satisfactory in many cases. The value of the closed circuit test is its ability accurately to predict energy requirements in closed circuit grinding for both homogeneous and heterogeneous materials. If the material is homogeneous, the open circuit test gives satisfactory results; but if the material contains appreciable fractions of hard and soft grinding ore, the open circuit tests will not be accurate because of the accumulation of hard grinding material in the circulating load. Since in most cases it is not possible to determine a priori whether the material contains hard and soft fractions, the closed circuit tests are preferable and more reliable.

B. S. Crocker *(Lake Shore Mines, Ontario)*—Dr. Roberts probability theory of grinding is very similar to our log pct reduced vs. log tonnage method of plotting and evaluating grinding tests at Lake Shore. However, although we both seem to start at the same point we finish with different end results.

Shortly after publishing our grinding paper (referred to by Dr. Roberts) in 1939, we did pursue the subject of the "constant pct reduction in the pct +28 micron material for each constant interval of time. We ran innumerable tonnage tests on the plant ball mills, rod mills, tube mills with 1¼ and ¾ balls, and lastly pebble mills, with tonnage variations from 180 tons per day to 950 tons per day. We found that when we plotted the log of the tonnage against the log of the pct reduced of any reliable mesh, we had a straight line up until 90 pct of the mesh is reduced. We have also tested this in our 12-in. laboratory mill with the same results. We have used this method of evaluating grinds for the past 8 years and developed the recent four stage pebble plant on this basis.

By pct reduced we mean the percentage of any given mesh that is reduced in one pass through a mill at a given tonnage (or time). For example, if the feed to a rod mill is 90 pct +35 mesh and the discharge at 500 tons per day is 54 pct +35, the pct reduced is $\frac{90-54}{90} = 40$ pct. If the feed had been 80 pct +35 the discharge would have been 48 pct +35 or pct reduced $\frac{80-48}{80} = 40$ pct as long as the tonnage remained constant at 500 tons per day. Thus we can easily correct for normal variation of mill feeds. This log — log relationship derived from the tonnage tests of all our operating mills has proved of tremendous help in checking laboratory work and in designing alternate layouts or new plants.

The difference between the log — log and the semilog plot is only shown up when the extremes in tonnages are plotted.

When the relationship between the pct reduced and the tonnage was first investigated, we used semilog

paper and apparently had established a straight line relationship between 500 and 800 tons per day on the ball mill and rod mills. However when the points below 500 (down to 190) and above 800 (up to 930) were plotted on semilog paper the two extremes were definitely not on the straight line on semilog paper but all points were on a straight line on log-log paper. We therefore, changed to the log-log plots and subsequent work has confirmed its reliability. Table VIII and Fig. 5 shows a recent tonnage test on our rod mill. The pct reduced for all meshes are plotted in this case. Usually only one or two key meshes are worked on.

We agree with Dr. Roberts that quick reliable work can be done in small laboratory mills. We have used a 12 in. diam by 7½-in. long laboratory mill since 1934 to prove all plant changes done on −8 mesh material. This includes ¾-in. ball testing and the recent pebble grinding with screened ore.

E. J. Roberts (author's reply)—It would be highly desirable if Mr. Crocker could find the time to expand his most welcome discussion into a full length paper. The writer has not had opportunity to make an exhaustive comparison of the two methods of handling grinding results; but, applying Crocker's method to the original data of the paper under discussion, it is found that only rarely do straight lines result by log-log plotting. They are straightest on the coarser meshes of another Kirkland lake ore (Ore C), but on Ore A all of the plots are decidedly concave toward the low tonnage—low percent reduced corner of the paper.

It may be that the use of both methods of plotting would prove fruitful in getting the most out of a set of data. One objection the writer has to the log-log method is that the sensitivity of the plotting is greatest where the screening accuracy is the least and vice versa. To illustrate; at high tonnages, the percent reduction is low; and the accuracy of this figure is low because it is obtained by taking the difference between two relatively large numbers. Yet on log-log paper, the difference between 9 and 10 pct reduction is over ¼ in displacement on single cycle paper. On the other hand at low tonnages and high percent reductions, the error as a result of screening may well be only about 50 pct of the other error; but the corresponding displacement even for the same error would only be about 1/32 in. (between 89 and 90 pct).

The above may not be the whole story because the effect of other errors should also be considered.

Mr. Bond is in error on both points of his second paragraph. The wording of the abstract describes a rate process and therefore power and not energy is the proper word. For the same reason, through the paper whenever hp/ton is mentioned, hp/ton is meant.

The writer was reluctant to use the term probability theory because of possible confusion with error curves etc. However, the basic thought of the writer's probability theory is the same as that of the first one of the probability theorems mentioned by Daniels[6] "The probability of an event depends on the number of possibilities."

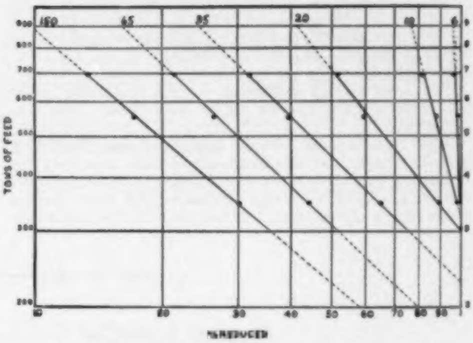

Fig. 5—Tonnage tests, 2½-in. rods, No. 5 rod mill, Nov. 6-29, 1950.

The statement that the closed circuit results showed no significant change in grinding behavior was meant to convey the thought that eq 4 still held and therefore there was no build up of hard particles.

Table I clearly shows that the higher the moisture content, the coarser the product of a 2-min. grind; and, therefore, the energy input required to obtain a given product must *increase* with moisture.

The reason for the low figure of 7.32 hp/ton of balls in the laboratory mill is the small diameter of the mill. Other things being equal, the hp per ton of balls increases as the square root of the diameter, see ref 4.

The C_{avg} corrections may well make the plant laboratory ratio farther from unity after correction than before. There is nothing sacred about unity here except for 2½-in. balls. The ratios for other ball sizes will naturally have to be different from unity.

In deriving any relationships regarding the effect of ball size on power from Table VI, the last two ratios (0.76 and 0.60) must not be used, because the data are heterogeneous with respect to the rest of the data. This is because in these last two cases the laboratory feed was 20 mesh instead of 4 mesh. This makes a significant change in the laboratory K's. The rest of the data correlate reasonably well on the formula:

hp ratio plant/lab = 0.73 (makeup ball diam) 1/3, eq 15

which is the relationship we use. However, as stated in the paper the data are neither sufficiently extensive nor accurate to permit any firm deductions on this point.

Table VIII. Screen Analyses—Tyler Standard Screens, 2½ in. Rods, Tonnage Tests, No. 5 Rod Mill

Tons Per Day / Tonnage Range / No. of Tests			352 Tons / 337 to 366 / 5			560 Tons / 530 to 590 / 2			697 Tons / 690 to 706 / 4		
Openings In.	Openings Mm	Mesh	Feed, Cum. Pct	Disch., Cum. Pct.	Pct Reduced	Feed, Cum. Pct	Disch., Cum. Pct	Pct Reduced	Feed, Cum. Pct	Disch., Cum. Pct	Pct Reduced
0.371	9.423		0.7						0.7		
0.263	6.680	3	15.0			15.8		100.0	17.0		100.0
0.185	4.699	4	35.2		100.0	35.5	0.2	99.4	37.2	0.3	99.1
0.131	3.327	6	49.9	0.1	99.8	51.2	0.9	98.0	52.3	1.7	96.7
0.093	2.362	8	59.9	0.4	99.4	61.5	3.1	94.8	61.7	5.6	91.0
0.065	1.651	10	68.2	0.9	98.5	69.7	8.2	88.2	70.6	13.2	81.2
0.046	1.168	14	74.9	3.3	95.7	76.0	20.1	73.5	76.9	27.1	64.8
0.0328	0.833	20	79.4	8.6	89.2	80.0	32.9	58.8	80.9	39.5	51.2
0.0232	0.589	28	82.5	17.6	78.6	82.8	43.7	47.2	83.9	49.5	41.0
0.0164	0.417	35	85.5	29.9	65.0	85.7	52.3	39.0	86.4	58.6	32.2
0.0116	0.295	48	87.5	40.4	53.8	87.5	60.8	30.5	88.1	64.9	26.4
0.0082	0.208	65	89.3	50.0	44.0	89.2	66.0	26.0	89.8	70.4	21.6
0.0058	0.147	100	90.8	57.8	36.4	90.7	72.0	20.6	91.1	74.9	17.7
0.0041	0.104	150	92.1	64.1	30.4	92.0	78.2	17.2	91.9	79.6	13.4
0.0029	0.074	−150	100.0	100.0		100.0	100.0		100.0	100.0	

If eq 15 is correct, the relative energy input is unity for a 2.56 in. ball, and, for what it may be worth, the work accomplished by a ton of balls per unit of energy input is proportional to the cube root of the ball surface area.

For open circuit estimates, we do essentially what Mr. Bond suggests, if the writer understands him correctly.

With regard to Mr. Bond's last paragraph, the writer merely stated that the batch grind data was used for obtaining a relative grindability series. In practice these figures are used for rough estimates. For more precise estimates, a closed circuit test is made at the particular mesh desired. The greatest danger in using these batch figures for closed circuit estimates comes from the effect of classification on minerals of different densities. Pyrite in the overflow of a classifier is much finer than quartz and as a result the average feed to the ball mill has a different ratio of pyrite to quartz than that of the raw ore and the grindability may be different. Also the last column of Table VII will no longer apply.

[5] F. C. Bond: Standard Grindability Tests Tabulated. *Trans.* AIME (1949) 183, p. 313. Mining Technology (July 1947). TP 2180.

[6] *Mathematical Preparation for Physical Chemistry*, Farrington Daniels, McGraw Hill Book Co. Inc. (1928).

Progress Report on Grinding at Tennessee Copper Co.

by J. F. Myers and F. M. Lewis

DISCUSSION

L. E. Djingheuzian *(Canadian Dept. of Mines and Technical Surveys, Ottawa)*—In their Summary the authors say: "Reconciling the grinding efficiency with good metallurgy is still a problem."

In the discussion of the first paper[3] in his reply to W. I. Garms, Mr. Myers states:

"Our grinding process with smooth 1-in. balls has reduced by nearly one half the metallic losses in the fine micron sizes of the tailing. This is simply because less of the fine micron sizes are produced. Since the + 65 mesh size is the same as formerly, a higher percentage of the intermediate sizes are developed. These sizes have the *highest floatability*, require the *least reagents*, and use *less floating time*.

"These factors contribute so heavily to the overall economies that dropping our power grinding gain from 28 pct back to 19 pct is a small detail. However, we feel that this is only a momentary situation and that eventually the best features of the grinding and flotation processes can be brought together, which is as it should be."

Italics are mine.

The above statements, to me, appear to be the answer to the opening statement in the Summary.

Denoting the costs at different power grinding gains as:

	Power Grinding Gain, 28 Pct	Power Grinding Gain, 19 Pct
Cost of grinding	G	G_1
Cost of flotation	F	F_1
Value of metallic losses	T	T_1

where $G_1 > G$, $F_1 < F$, and $T_1 < T$, we have:

$$G_1 + F_1 + T_1 < G + F + T.$$

Since the authors accept the idea that "grinding in flotation plants becomes part of the 'conditioning' of the feed to flotation",[4] i.e., that in flotation the ball mill is primarily a conditioning machine, it can be postulated that Tennessee Copper grinding at cost G_1 is more efficient than grinding at lower cost G. This can be directly inferred from the Conclusion of the paper. Mr. Myers also emphasizes this at the end of his reply to Mr. Garms: "that grinding is for the purpose of preparing flotation feed and not grinding per se."

This, to me, in the final analysis means that when the efficiency of grinding is weighted against the conditioning factor, the former becomes a function of efficient conditioning, hence, within the system in which proper conditioning is the dominant factor, the best grinding efficiency is provided by grinding which will contribute towards the optimum conditioning. This brings us again to the statement: "that if every grinding unit were considered as a conditioner for each following step, efficient grinding plants would become much easier to design."[4] In other words, grinding equipment should be balanced against the flotation equipment and against chemical reactions taking place in the system.

F. C. Bond *(Allis-Chalmers Mfg. Co., Milwaukee)*—The authors' discussion of the probable ball motion in a slow speed high dilution mill is very interesting. When the 1-in. balls have worn down to about one fourth of their original weight they apparently first develop a flat surface; as wear progresses this flat face becomes concave, and other concave faces appear. It seems more probable that the first flat face may form at the softest part of the ball surface, and that each succeeding contact tends to force this flat face into sliding contact with a larger round ball; than that the flat faced ball tends to pair off with a particular round ball and to travel with it continuously. When the small worn ball has a flat face and is in sliding contact with a large round ball, the surrounding large balls will assume a more or less definite pattern, and slide against the worn ball, thus producing secondary concave faces. The primary concave face seems to be larger and better developed than the secondary faces.

The ball charge can be divided into "concaves" which show at least one concave surface, "intermediates" which have developed flats or incipient concaves, and "rounds."

Ball slippage is always present in a tumbling mill, and the mutual ball movement is necessarily a combination of rolling and sliding. The sliding motion is apparently concentrated upon the smaller worn balls which nest between the surrounding larger round balls. When each worn ball starts its upward path in the mill its primary flat or concave surface fits against a larger round ball, and the round ball slides upon it. The action may be something like that of the ball separator in a ball bearing, except that the worn sliding balls are always under considerable pressure.

The material is ground under the combined influence of breakage 1—by impacts between falling balls and between falling and supported balls, 2—by being nipped between rolling balls, and 3—by being rubbed between the sliding balls. The rubbing action will be increased in the presence of worn balls with concave surfaces. The rubbing action probably produces a considerable portion of the finely ground slimes in the product.

The worn balls commonly approach tetrahedrons in shape, and are very different from concavex, each of which has two equal opposed concave surfaces. Concavex were designed only to grind upon themselves, and not for use in combination with grinding balls. Their action in a grinding charge is very different from

that of the concaves, or small worn balls with concave surfaces.

Laboratory grinding tests should give an indication of the change in efficiency caused by the presence of the concaves. A series of such laboratory tests has been planned.

There can be no question but that grinding efficiency should be considered in the light of overall plant results rather than from the standpoint of the size reduction accomplished. However, no convenient yardstick other than size analyses is presently available by which the effect of a proposed grinding installation upon the plant recovery and reagent cost can be predicted. Until such a yardstick is developed, it will be necessary to consider grinding primarily on the basis of feed and product sizes.

[3] J. F. Myers and F. M. Lewis: Progress Report on Grinding at Tennessee Copper Co. *Trans.* AIME 187, (1950) p. 707; *Mining Engineering* (June 1950). TP 2863B.

[4] L. E. Djingheuzian: A Study of Present-Day Grinding. *Trans.* C.I.M., Vol. 52: 243-257 (1949).

Flotation Rates and Flotation Efficiency

by Nathaniel Arbiter

DISCUSSION

T. M. Morris *(School of Mines and Metallurgy, Rolla, Mo.)*—Rate studies promise to help quantify flotation operations. The author's exposition of rate studies is therefore laudable. However his exposition of a second order rate equation rather than a first order rate equation is not convincing. It is important to use the correct equation, otherwise erroneous conclusions may be drawn.

The data assembled by the author and plotted according to a second order rate equation can be plotted according to a first order rate equation. As a matter of fact much of the data used reveals that a second order rate equation is not applicable. According to the method used by the author for plotting data, the reciprocal of the slope of the line is the maximum possible recovery. Physically this cannot exceed 100 pct, yet several graphs yield maximum recoveries greater than 100 pct. For example: 1—Curve 2, Fig. 1 yields a value of 120 pct; 2—Curve 2, Fig. 2 yields a value of 115 pct; 3—Fig. 7 yields a value of 110 pct.

The real test of the graphical method is, as the author points out, whether or not the earliest points agree with a straight line. The second order rate plot does not fulfill this requirement. Fig. 10 shows the data used by the author to obtain Curve 2 in Fig. 1, plotted according to both a second and first order rate equation. Fig. 11 shows the two rate plots made from data published by Ludt and DeWitt.[10]

In Fig. 10 the first order rate plot gives a very good fit and obeys the boundary condition of passing through the origin. For the second order rate plot a straight line can be drawn through the last three points but the *earliest* points are not in agreement with this line. The reciprocal of the slope of the straight line gives a value of maximum possible recovery of 112 pct. The maximum possible recovery according to the first order rate plot is 78 pct.

In Fig. 11 the first order rate plot again fits all of the points very well and passes through the origin. The second order rate plot shows wide deviation of the *earliest* point, indicating that the second order rate equation is not applicable.

The reason for the *fit* of data to the second order rate equation during longer flotation times is that R, the cumulative percent recovery, changes slowly toward the end of the flotation operation, so that when t/R is plotted against time, one is, in effect plotting t against t and one would certainly expect a straight line.

The author attempts to show that the second order rate equation is correct by citing evidence which shows that recovery is a function of initial concentration. This is a debatable point especially since, in the author's original presentation of this paper at the St. Louis

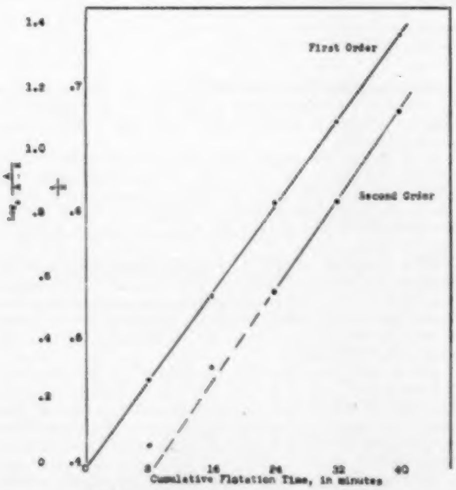

Fig. 10—Data of Zuniga, gold tailings.

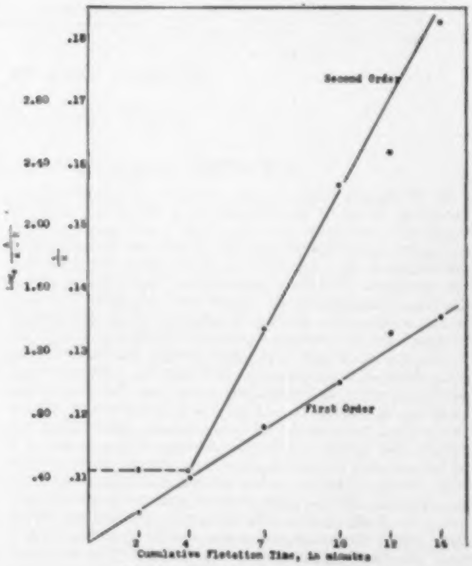

Fig. 11—Data of Ludt and DeWitt.

meeting of the AIME, he included a table showing that recovery was essentially the same when the pulp density varied (which is similar to variation in initial concentration).

We have been conducting rate studies at the Missouri School of Mines and Metallurgy and find that a first order rate equation is applicable to our data.

Nathaniel Arbiter (author's reply)—The criticisms raised by Mr. Morris consist of the following: 1—that for one out of 18 sets of data in the paper, and for one additional set, a first order equation gives a better fit than a second order; 2—that for three of the sets of data, one of the constants in the equation does not have real physical meaning.

Granting the correctness of the criticisms, I do not see that they affect the thesis of the paper. The paper does not advance the second order rate equation as exclusively applicable. Rather, it seeks to demonstrate the utility of the rate approach in understanding the flotation process.

[10] R. W. Ludt and C. C. DeWitt: The Flotation of Copper Silicate from Silica. *Trans.* AIME (1949) 184, p. 49; *Mining Engineering* (February 1949). TP 2535B, Table I, Experiment No. 36a, b, c, d, e, and f.

A New Method for Recovery of Flake Mica

by R. Adair, W. T. McDaniel, and W. R. Hudspeth

DISCUSSION

O. C. Ralston *(U. S. Bureau of Mines, Washington, D. C.)*—Flake mica can be beneficiated by a wider variety of methods than almost any known mineral. However, most of these methods are not recorded. It is therefore a pleasure to find a written record of another application of a new method of beneficiation.

The fact that the Humphries spiral gives a concentrate containing the mica flake mixed with the slime and small grit has forced a further separation by screening, in which the finest of the flake will pass through. Fine flake has its uses and if mixed with slimes or clay is recoverable reliably only by flotation. This loss must be charged against the spiral. On the other hand, the new muscovite mica paper recently brought out by General Electric Co., can be made only with grit free flake larger than 50 mesh. It is of interest to learn from the authors whether the washing of the flake on a 50 mesh screen can take the grit content below 1 pct, to meet this new demand. Must another degritting process be used on the flake, like electrostatic treatment of the dried material?

H. D. Snedden *(The Humphreys Investment Co., Denver)*—When recovering mica in the spiral, gangue particles coarser than 16 mesh can be rejected more readily when operating at a high pulp density (35 pct solids) or a low volume rate (—15 gpm).

A launder screen as used by Lehigh Coal Co. has proved very satisfactory to partially dewater the mica concentrate and to reject any gangue slimes it may contain.

Under certain conditions mica cleaning in the spiral can be improved by reagentizing the spiral feed, the object being to insure wetting of the gangue minerals and to agglomerate or prevent wetting of the mica particles.

R. B. Adair (authors' reply)—The grit content of the mica recovered on Humphreys Spirals varies with the fineness of the launder screen used to clean the mica product. Normal practice has been the use of 80 mesh screen, which will produce a mica product of acceptable grit content for most purposes. By using a 50 mesh screen, and increasing the screen area per ton of feed, it is felt by the authors that a mica of 99 pct plus purity could be obtained. This statement would hold only in the case of mica feed which was well delaminated, and free from "clay balls," (semikaclinized feldspar), which tends to follow the mica through the process.

Flotation Tests on Korean Scheelite Ore

by Will Mitchell, Jr., C. L. Sollenberger, and T. G. Kirkland

DISCUSSION

R. S. Handy *(Santa Rosa, Calif.)*—It would be interesting to learn the comparative results of treating the Korean scheelite ore described in this paper according to the following procedure: 1—Follow the procedure described in the paper to the recovery of the sulphides by flotation. 2—Dilute the residue from the sulphide flotation strongly with water and disperse the contained colloids by adding a mixture of 75 pct sodium silicate and 25 pct quebracho to dispersion point. 3—Settle the dispersed pulp and decant the supernatant colloids. 4—Dilute, settle, and decant the pulp until the settled residue is crystalline. Not over 10 pct of the tungsten minerals should follow the colloids. 5—Treat the settled residue by flotation, having established the proper pH. Oleic acid homogenized with cresylic acid or kerosene is more selective for scheelite than is oleic acid alone. (The recovery of the contained tungsten minerals should be practically complete in this operation.) 6—Deflocculate the tungsten concentrate from step 5 with the same dispersion agents as in step 2 and treat the deflocculated pulp by gravity. The minerals that tend to float with scheelite are generally of low specific gravity and the separation is efficient. The heavier contaminants, such as pyrrhotite and magnetite can be removed magnetically, leaving a practically pure scheelite.

E. J. Pryor *(Imperial College of Science and Technology, London)*—It occurred to me that instead of using temperature as a differentiating control, more precision might be obtained with a reagent of a type we developed here for the flotation of high grade fluorspar in cold pulps.

This reagent, which is manufactured on a modest scale in England, is based on oleic acid, which has been modified by our patented process to ensure complete stable dispersion at any desired concentration in water. It thus becomes possible to utilise unimolecular layers with good selectivity.

In the fluorspar flotation for which we developed it, it is possible to hold a tight control on depressing quebracho for the calcite without substantial loss of fluorite, and recovery exceeding 99 pct is not uncommon in the mill where it is in use, even under wintry conditions.

No attempt has been made yet to extend the use of the reagent, as we have not completely overcome

manufacturing and packaging difficulties, but it has shown good results on numerous fluorspars from various parts of the world, and we are extending the tests to other nonmetallics now floated by straight oleic acid. It is not easily upset by hard water, though for laboratory tests distilled or deionised water is desirable in the first place.

If you wish, I will send you a small free test quantity. Until we have succeeded in making it as a solid, commercial application seems to us limited. Cost should ultimately vary up to 30 pct above that ruling for oleic acid.

Will Mitchell, Jr. and **C. L. Sollenberger** (authors' reply)—The purpose of this investigation was to determine the effect of several variables encountered in nonmetallic flotation on the recovery and grade of scheelite in rougher concentrates and did not intend to imply that the combinations used were considered ideal for maximum recovery and grade in cleaned concentrate.

We have investigated the procedure suggested by Mr. Handy and have found that about 10 pct of the scheelite was lost in the desliming operation. With a sodium silicate-quebracho mix a recovery of 76 pct of the scheelite was obtained in rougher float at a grade of 18 pct. With additional depressant in the cleaner float, an increase in grade to 22 pct was obtained but much of the coarse scheelite did not float; the recovery in this operation dropped to 36 pct. The final cleaned flotation concentrate from the procedure consisted mainly of fluorite and scheelite. Since the ratio of these minerals in the head sample was about 2 to 1, the same ratio was obtained in the cleaned concentrate. This concentrate would have to be further treated for removal of fluorite by flotation as described in the paper or by tabling as suggested by Mr. Handy.

Mr. Pryor kindly sent us a sample of the reagent mentioned and we tried it on the ore. No spectacular results were obtained although the reagent was a good collector for scheelite. Unfortunately the ore contains about twice as much fluorite as scheelite and the reagent does not aid this separation.

F — Coal

Coal Preparation in England and Holland

by John Griffen

DISCUSSION

K. F. Tromp *(Kerkrade, Holland)*—Your assumption that the Dutch State Mines have lead in the development of heavy medium processes—the Barvoys, Loess, Driessen—is not correct. The credit should be given to a Dutch engineer Mr. deVooys, the inventor of the Barvoys process.

deVooys started in 1932 in Hückelhoven, a German but Dutch-owned mine, where he was a director. Several plants on the Continent as well as in Great Britain were erected.

The State Mines started their first plant in 1937. They could not get that plant working satisfactorily and built another one. Their process made no headway, and up to 1946 there was only one plant installed, the one at their own pit. Since 1946 they started the cyclone process and only since that time have they been more active, though industrial cyclone washeries are still in their infancy.

deVooys sold his patents to the Dutch State Mines in 1946 and this might have given you the impression that the pioneer work had been done by the Dutch State Mines.

John Griffen (author's reply)—I wish to thank Mr. Tromp for giving the correct story on the development of heavy medium processes in Holland.

Factors Influencing the Choice of a Loading Machine

by D. W. Mitchell

DISCUSSION

J. H. Schlobohm *(Joy Manufacturing Co., New York)*—This paper has been read with a good deal of interest; however, there are several salient features which Mr. Mitchell has overlooked.

The initial emphasis on selection should be placed on type of material to be loaded and method of mining involved.

There is no possible connection in the comparison of scraper or Joy Lohite loaders with loaders of any other type. Neither can an operating comparison be made between "Rocker type" loaders and those equipped with loading head and gathering arms. Each unit has its application, but the application depends on material and the mining system involved.

Mr. Mitchell's compilation of data on various loaders, while it represents a great many facts, does not represent true operational data, i.e. Table I, column heading "Max. Rated Capacity for 7 hr;" these figures are meaningless inasmuch as little or no time at all is allowed for tramming or switching or moving the loader from place to place. Table I, column heading, "Best Reported Field Shift," would be a better source of operational data, assuming of course that conditions, applicable to each class of loaders, were average to avoid enthusiastic performance figures.

Table I, column heading, "Repair Parts Cost per Ton," is still another item which will not stand up under comparison unless the loaders involved are put into type classes, i.e., there are simply no means possible to show the repair costs of scraper loading versus a caterpillar mounted loader in abrasive material. The source of information here is also important. Actual field tests have proved that these costs are of a constant nature. Manufacturers are striving to hold them to a minimum and many interesting studies have been made. A fairly recent example of this in a large United States gold producer has proved that Joy HL-3 loaders can be consistently average, less than $0.025 per ton mucked. This cost represents repair labor and parts over a two year

period in heavy ore. Again, however, no real comparisons of repair costs can be made without taking the actual working conditions into consideration. It has been found repeatedly that many operators are perfectly satisfied with high maintenance costs to achieve required tonnage. These high costs may be caused by poor bottom, power conditions, or even minor maintenance neglect but all of these factors are justifiable if the end is accomplished. This is particularly prevalent on high speed contract work.

Many of the loaders listed are typical "hard rock" loaders, particularly the Joy Model 18HR2 and the Goodman Conway loaders. The Joy 18HR2 has a rated capacity of 6 to 12 tons per min and is by far the fastest loader available on the market. This machine is now being successfully used in many mines in the United States as well as abroad and while we know that tonnages of over 900 tons per shift have been mucked, average tonnages run about 500 to 550 tons per shift. These figures are based on tonnages of iron ore and would not be applicable in high speed tunnel work where the Joy 18HR2 is perhaps best adapted. Here, in large bore tunnels where deep rounds can be pulled, this machine is chalking up remarkable tonnage records.

Based on the aforegoing it is felt that a more realistic view of the problems be taken into consideration before any actual choice of loading machine be made. The operator should under any circumstances arrange to see loaders working in conditions which are similar to his own. Then, after careful study, he should know which machine is best adapted to his particular circumstances. Consideration should be given to the factors influencing the choice of a loading machine in the following order:

1—Material to be loaded, 2—Mining system, 3—Required tonnage, 4—Maintenance cost, 1 + 2 + 3 + 4 = *class of loader* to be used.

It is granted that there are many more factors and even subfactors, but these basically should be given primary consideration. Regardless, however, of what selection is made, a mine loader is only as efficient and productive as operating conditions permit. Transportation of material as well as location of working places are too often neglected factors in loading problems and without proper auxiliary equipment full productive power cannot be reached.

In the equation above the term *class of loader* is italicized. The operator is now ready to take into consideration such factors as maximum dimensions, power means, type mounting, etc. The importance of these features is minimized because it is felt that a selection can be made from the wide range of available machines, and to sacrifice the advantages of a minor factor will in all probability lose its importance when high tonnages are attained. The manufacturers are more than willing to consider special applications and in most cases standard loaders of one class or another can be utilized. In any event, the operator will do well to consult with manufacturers or their representatives concerning problems which involve machinery.

Efficiency and Sharpness of Separation in Evaluating Coal-Washery Performance

by H. F. Yancey and M. R. Geer

DISCUSSION

John Griffen *(Pittsburgh)*—I wish to congratulate the authors on this paper, which, I am sure, will promote a clearer conception of the various criteria which have been advanced as measures of coal-cleaning performance. Their clear descriptions of the more important criteria that have been advanced and explanations of their meaning and application have been sorely needed and, if widely promulgated and followed, should remove much of the confusion which, heretofore, has been evident.

I suggest that much of this confusion has also arisen from lack of common agreement as to the exact meaning of the words and terms used in our discussions of this subject. I am sure I have been equally guilty in this respect and suggest it is time that we select a committee to study vocabulary as well as criteria, so that common understanding and agreement can be had.

Thus, I suggest, that the title of this paper should have used the words "coal-cleaning performance" and not "coal-washery performance." My understanding is that "washery" refers to a complete plant and it has additional functions to that of coal-cleaning, such as control of size, moisture, etc.

Also, I do not agree with the authors that the formulas given in their paper, the Fraser and Yancey formulas, are necessarily the only ones that can be called efficiency formulas. When a dictionary is consulted one realizes that the words "efficiency" and "efficient" have not precise meanings but only very general ones and must be further defined for each particular use, if they are to have any precise meaning.

To my mind, the first of the Fraser and Yancey formulas given should never be used as expressing "efficiency" alone but only by the fuller phrase which truly expresses the kind of efficiency it states, which is "coal recovery efficiency." For complete clarity we then need only define "coal" for the particular circumstances under which the formula is used. This would recognize the fact, fully explained by the authors, that the limits in coal qualities imposed by the market have imposed a factor—and a variable factor—which has influenced any answer obtained by this formula. Such a procedure makes it entirely practical and logical to use the values of any of the qualities of coal—ash, sulphur, phosphorus, specific gravity—or a combination of them to define what is coal for that particular use of the formula.

Further, it is clear to me that the criteria to be used must be dictated by the objective in view and the use for which the data are obtained. The authors clearly point out that the Fraser and Yancey coal-recovery efficiency formula is influenced by three factors, "one is determined by the coal, one is dictated by market or use conditions, and one is an inherent characteristic of the cleaning unit itself." The formula which uses the inverse of misplaced material as a measure of efficiency is also a coal recovery efficiency formula and is influenced in like manner by these three factors. As the authors point out, both are useless in obtaining data which enables a direct and valid comparison of the performance of a given cleaning unit on two coals where the other two factors vary or of the performance of two or more cleaning units of different types unless the values of the other two factors are identical. Much of the confusion in our discussions of and literature on coal cleaning has come from failure to understand clearly this fact. Many such comparisons have been made from which erroneous conclusions have been drawn.

I am, therefore, particularly glad to see the authors' emphasis on the criteria which indicate the "sharpness of separation." I feel that these are the essential criteria

in which the coal cleaning fraternity should be primarily interested—technically and practically. The characteristics of the raw coal were determined by nature and the mining organization, while the qualities of the washed coal are dictated by market requirements and the opinions of the coal sales department. These two factors are seldom subject to change. Thus, the sharpness of separation of the various types of coal cleaning equipment is the only one of the three factors over which any control can be had. It is only by the proper selection of cleaning units having the necessary sharpness of separation to produce the product the market requires from the raw coal available, that the optimum coal-recovery efficiency can be obtained. The use of the coal-recovery efficiency data as the only guide in selection, when the other factors are dissimilar, confuses the issue and has resulted in unhappy selections.

Even the use of the various criteria to determine sharpness of separation seems to need further exploring before they can be used as precise tools. We need far more data on these criteria for each type of coal cleaning unit, which shows the effect of size range and of variations in throughput, as well as the gravity of separation, on the sharpness of separation. In like manner we need data which will indicate the characteristics of each type of cleaning unit as to the cleaning performance on individual sizes of coal of a wide size range being cleaned in one unit. This is of vital importance when such individual sizes are to be marketed as such.

It has been shown by the authors and others that the various coal-recovery efficiency formulas are ineffective for the purposes described in the preceding paragraph and only the criteria which evaluate density and sharpness of separation are useful. Therefore, it seems to me the authors beg the question when they stress the greater amount of test work, including a wider-range of testing densities reaching to higher gravities, required to evaluate properly these criteria. It is only by conducting our investigations thoroughly and accurately that we can arrive at valid conclusions. Therefore, I feel confident that the test work to be done for these purposes must be safeguarded by adherence to all the details which the Europeans have found essential. It is my understanding that the most important of these conditions are:

1—Distribution curves and any criteria derived from them must be based on a reconstituted feed, to eliminate any errors arising from degradation or crushing of the feed material during treatment and from sampling inaccuracies.

2—A sufficiently wide range of testing gravities must be used so that the distribution of the coal of lowest gravity and the refuse of highest gravity can be determined within a probability of error that would be insignificant. I believe K. F. Tromp claims that the values obtained for these criteria are questionable, unless the distribution curve can be closed at both ends.

I offer as an additional suggestion that the reporting of distribution data on a fine size reaching to zero is very questionable, certainly for equipment utilizing gravity as a means of separation. We feel quite sure that at some small particle size differences in specific gravity become an insignificant factor in effecting separation. Further, on such fine materials, the separations made by the testing liquids are far from accurate unless centrifuges and elaborate retreatment are employed.

Referring to the subject matter of condition 1 above, I would say that failure to observe this requirement has been most glaring in this country and has resulted in the presentation of performance data which is very definitely misleading. The paper presented by R. E. Zimmerman[17] contains an illustration. In his Fig. 8 he plots the cumulative float ash data of the feed sample and of the cumulative table zone products. A comparison of these curves shows a very high coal-recovery efficiency for separations at 1.50 sp gr and higher densities. However, the feed sample data shows a total of 20.8 pct ash while the total of the table zone products is only 19.5 pct ash. The products were tested at only 1.5 and 1.6 sp gr and a calculation of the reconstituted feed at these two gravities shows a wide divergence from the amounts and ash contents of the feed at these points. When the cumulative zone products are compared with the reconstituted feed, the amount of misplaced material is doubled and the coal-recovery efficiencies in this ash range of clean coal are very significantly reduced.

As the authors point out, the error area may not necessarily have a fixed relation to coal-recovery efficiency. I suggest, that if a system be devised that will properly weight the respective error areas according to their distance from the density of separation, a term could be obtained that would bear a much closer direct relation to coal recovery efficiency. Whether it would provide a more useful criterion is uncertain.

The definition of the specific gravity of separation as that at which the distribution curve crosses the 50 pct line if considered alone may result in wrong assumptions. On page 514, the authors point out that the wet table distribution curves for different size ranges show a trend opposite to that of jigs, in that, "with the wet table, decreasing particle size is accompanied by decreasing density of separation." Reference is made to Fig. 8 which confirms the statement as made. Normally, one assumes that a separation at a lower density should result in a washed coal of better quality. However, if one examines these distribution curves, one notes that a feed size range of 8 mesh to 0 has been examined in three sizes, 8 to 20 mesh, 20 to 48 mesh and 48 mesh to 0. The density of separation, as defined, progressively decreases but the error area of the 48 mesh to 0 has increased so much that it is very likely that this size of the washed coal is higher in ash than either of the other sizes. It is probable that the 20 to 48 mesh washed coal, which shows an intermediate density of separation, has the lowest ash content and shows as sharp a separation as any of the three sizes.

In conclusion, I feel our situation is similar to that of the statistical presentation of data, where no single term, such as average, average deviation, etc., fully defines and represents the facts; but a combination of a number of terms is required. To provide tools which we can use to evaluate coal-cleaning performance properly, we also need several terms or yardsticks, so that we may advance the art of designing and operating such plants to obtain optimum results with minimum costs.

I feel the authors' paper will prove to be a most important guide to a proper appreciation of the conceptions so far advanced and in educating the coal industry to their proper use and they are to be highly complimented. The crying need is for complete and accurate test data in great wealth which can be evaluated by these various yardsticks so the capabilities of coal cleaning equipment can be known under all conditions. Coal has become too valuable to be wasted by the improper application of equipment which under different conditions has done an efficient job.

W. W. Anderson *(Commercial Testing & Engineering Co., Chicago)*—Messrs. Yancey and Geer have prepared an exceptionally thorough and scholarly review of the better measures for evaluating coal washery performance. In addition to presenting discussion of the usefulness of various measures of performance, the paper contains several important conclusions not found elsewhere in American literature, particularly in regard to the worth of error areas and probable errors.

Apparently a number of investigators have been motivated by the casual resemblance of the curves of distribution to the Ogive, or cumulative frequency curve of statistics, to hope that distribution curves would behave in accordance with the laws of probability. Numerous writers have stated that the shape of the distribution curve, with the top half inverted, is similar,

except for skewness, to the Gaussian error distribution curve, neglecting to mention that most distribution curves have positive kurtosis, or a peak at the gravity of separation greater than that of a normal curve of probability. Furthermore, the curves of distribution which neither look like the normal curve, nor any other smooth curve of frequency distribution, are usually disregarded. Therefore, it is not altogether surprising that a probability plot of distribution data fails to develop a straight line.

Nevertheless, the idea that the laws of chance may apply at least to some curves of distribution should stimulate efforts, not only to ascertain causes for deviations of distribution curves from probability frequencies, but also to characterize distribution curves by a few arithmetical constants or suitable formulae.

The lack of correlation between error area and misplaced material should not have been unexpected, since error area is obtained from the distribution curve which is a plot of ratios, whereas misplaced material is expressed as a direct percentage of all products.

It is clear from this paper that error area reflects only distribution of misplaced material, rather than the amount of misplaced material. A three-dimensional value, such as error volume, is needed to express quantity and distribution of misplaced material. After all, the surface area of a lake does not indicate the volume of water in the lake unless the additional factor of depth is known. Why any of us should have expected error area, as currently determined, to measure efficiency, or the amount of improperly distributed material, is more astonishing than proof to the contrary.

More than a year ago, W. L. McMorris of the H. C. Frick Coke Co. suggested to the writer that the abscissa scale should not be laid out in equal intervals of specific gravity, but in proportionate intervals of gravity, in accordance with weight distribution of the gravity consist of the combined products. This would appear to be a reasonable suggestion, not only to enhance the value of error areas, but also to present more perfectly the curves of distribution. The writer regrets that he did not have time to prepare evidence for this discussion of the worth of this suggestion; but the idea may have sufficient merit that other investigators would want to study it for themselves.

Plot of the data in Table IV of percent misplaced material versus efficiency indicates a pretty fair correlation, and a similar plot of the data in Table III demonstrates an even better correlation of these two measures of efficiency. Incidentally, it should be noted that Yancey speaks of his measure as "efficiency of recovery," whereas Anderson terms his measure "efficiency of separation," which in each case is proper terminology. Since values determined by either method appear to be of the same order, use of either method should not lead to confusing interpretations, except in the case where no true separation has occurred. In such a situation, Yancey would report 100 pct efficiency of recovery, and Anderson 0 pct efficiency of separation.

Undoubtedly there are occasions for the use of one method in preference to the other for rapid and inexpensive determination of washery efficiency. For instance, if the float and sink data on the feed are available, the recovery method is the quickest approach to efficiency. However, when feed data are not available, the most rapid method to ascertain efficiency is from the amount of misplaced material. This latter method is also the cheapest when appreciable size degradation occurs in the washing process, because then Yancey's method cannot be used safely without complete analytical data on the washery products for calculation of a composite product gravity consist.

Whenever Yancey's efficiency of recovery is applicable, the user should remember that Yancey's method tends toward high values of efficiency because the method credits size degradation to washery efficiency.

At some cleaning plants preparing metallurgical coal, where size degradation means no loss in sales realization, the high values of efficiency determined by the Yancey formula are probably of no great consequence. In contrast, many operators of cleaning plants specializing in preparation of stoker coals are vitally interested in not being misled concerning washery efficiency and size degradation, because these items directly influence sales realization.

A minor point to which exception is taken is the assumption as stated in the following quotation from the paper: "European practice apparently involves the use of more and higher densities in examining samples for performance tests. Thus, American practice may have to follow the European system . . ."

Clients of the Commercial Testing & Engineering Co. know that performance tests of their cleaning plants include a sufficient number of gravity fractions to properly locate the curves of distribution; and if fine sizes are under study, densities up to 2.17 are incorporated in the test procedure. Hence, any assumption that all American practice lags behind European methods is false.

In conclusion, the writer feels that American coal preparation engineers should be grateful to Messrs. Yancey and Geer for their excellent paper, which not only reviews the Paris meeting of the First International Conference on Coal Preparation; but in addition, and what is more important, it clarifies some of the measures of performance which have been in sad need of critical study and unbiased evaluation.

G. A. Vissac *(Seattle, Wash.)*—Coal preparation in the United States has shown remarkable progress and advances during the last few years. As a consequence there is now a real demand for more scientific methods of controls, and the paper of H. F. Yancey and M. R. Geer will be appreciated by all washery engineers.

It would be desirable to unify terms and methods. My own solution has been to adopt terms and methods now in general use in this country, in connection with other industries, sciences and activities.

There is nothing new in the mathematical analysis now applied to coal washing; use is made of the well-known laws and theories of chance, or great numbers, or probabilities, fully described in all technical books dealing with statistics.

Many terms such as dispersion, mode, frequencies, probable error are defined; I submit they might be adopted by the coal industry, just as they have been adopted by many other branches.

I suggest we should also adopt the same methods and standards of graphic presentation; namely, the straight line presentation commonly used for similar problems, and for which standard graph papers are available in this country (Codex Co., Norwood, Mass.).

The advantages of logarithmic scales to present such distributions are obvious. However, we must learn how to use, or adapt, these standards, methods and graphs.

First, tendencies only must be looked for; not accurate lines, but the nearest lines covering the events most likely to happen.

The mathematical laws we are using are actually called laws of great numbers, in other words, they will only best cover the most common results out of a large number of tests.

Tests must be kept within practical range; for instance, densities at 1.3 or 2.2 are of no practical value in most coal cleaning problems.

The value and purpose of the distribution curve, or line, is to evaluate as correctly as possible, but relatively, three main characteristics; namely, the actual gravity of separation and the two probable dispersions.

As the first characteristic is found at the 50 pct distribution, and as the other two are found respectively between 25 pct and 50 pct, and 50 pct and 75 pct respectively, it is obvious that the only interesting part of the distribution curve is between 25 and 75; and the points beyond are of very little value, and the main thing is to concentrate the tests on the results between the above critical points.

Origins and scales must be carefully selected. The most common errors in this connection are due to the

fact that sufficient distinction is not always made between what is measured and what is analyzed.

The ash content of a product is made of a constant: the inherent ash characteristic of this particular coal and size, plus a variable; the impurities, or removable ash; and this is the only variable to be considered if we are to use logarithmic representations correctly. This can be translated in gravities, namely, gravity of the pure coal, gravities of the foreign materials.

However, gravities must be considered as a "mean" only, but the final practical results must always be expressed in ash contents.

Pure coal, inherent ash must be understood at their practical value, namely, for each particular size of each particular coal, it will be the float at the low practical density; namely, from 1.30 to 1.40, or to be more precise, the density of separation less — 0.20, or — 0.15.

It is well known that the various sizes of the same coal have different sink and float characteristics. Furthermore, most of the new methods of control have recognized the influence of size on results obtained with a given machine and a given coal; still many tests used for demonstration purposes do not take sizes into consideration, and accordingly are of no practical value.

In all cases, to be able to compare results and to interpolate tendencies, a definite and narrow size ratio must be dealt with, and the ratio of 2 to 1 suggested as narrow enough to give consistent and scientific results; namely, the following sizes: 2 in.—1 in., 1 in.—½ in., and ½ in.—1.4 in.

Results of operation on such wide ranges as say ¼ in.—0 are not suitable to scientific analysis; and trying to compare an operation on say ¼ in.—0 size to an operation on say ¼ in.—1 in. is absolutely without value.

On the above scientific basis only can we hope to approach accurate and useful methods of control. Existing operations must be analyzed first to determine and analyze the parameters and characteristics involved.

They can be classified in two groups; 1—Factors proper to the washing machine when dealing with various sizes, various coals, and various densities, and 2—Factors proper to each particular size of each particular coal.

Obviously, the results of a great many tests must be analyzed and put in order in simple and workable forms, then translated the results in simple "empirical laws." As a result we will then be equipped to tackle the next problem, namely, the scientific design of a new preparation plant.

Only by the application of such scientific methods can we expect to clarify the situation as it exists today. New methods, new machines, are brought up (or unearthed); new and more stringent requirements are imposed; lower grades of coals have to be considered; it becomes more difficult to make a correct selection between the many solutions available today.

However, as a matter of fact, there is no such thing as a better machine, or a better process, but there may be a more desirable machine or process to suit each individual case.

On the basis of market requirements and characteristics of the raw product, a scientific approach only of the problems involved will supply the best and most economical solutions. We must be grateful to Cerchar and Yancey and Geer for their outstanding work in this connection.

Arthur Grounds and **L. W. Needham** *(National Coal Board, London)*—The authors have been careful to draw a distinction both in the title and in the paper itself between the sharpness of separation and the efficiency of the cleaning process in particular cases. It will be possible to make more rapid progress with regard to the proper evaluation of processes when this distinction is much more widely understood than it is at present. Attempts to express the sharpness of separation are, broadly speaking, attempts to express the characteristics of cleaning processes. The efficiency of separation, on the other hand, is an expression of the effect of the characteristics of processes on the products of separation when particular coals are treated.

The suggestion underlying many contributions to the subject, notably those by K. F. Tromp, is that the sharpness of separation characteristic of certain processes is, to a very large extent, independent of the composition of the raw coal feed. This suggestion is often received with considerable scepticism simply because it is not properly understood. Yancey and Geer, however, make very useful comments about it, especially on page 514 in the paragraph headed "Influence of Density Composition of Raw Coal." Clearly, if there are characteristics of processes which are independent of the density composition of the raw feed, and these can be understood, the results of separating any particular coal of known composition can be calculated beforehand with some accuracy.

Characteristics of processes may not be important if different methods of treating the same coal are compared because it may be sufficient simply to compare the results obtained. On the other hand, if it is desired to know how a given process would deal with a variety of coals, its inherent characteristics are of fundamental importance.

Yancey and Geer's paper is a valuable contribution to clear thinking on these points.

It is true that there is at the moment a lack of experimental evidence to decide how far characteristics of processes, independent of coal composition, can be justified. The experiments referred to in the paper should give very valuable information. In the meantime, it may be suggested, on the basis of experience with British plants, that the distribution curve is probably substantially independent of the composition of the raw coal for dense medium washers, especially those using a comparatively stable medium in which currents are at a minimum, but that in Baum or trough washers, where the bed or "separating medium" is made up of selected parts of the feed, it is much less likely that the distribution curve is independent of the density composition of the raw feed.

Fig. 4 is of particular interest and may be compared with the graph which we have prepared, Fig. 11, and which shows the relationship between error area and "écart probable" for a number of separations in British plants. The point shows a good distribution about a straight line as distinct from the curve as suggested by Yancey and Geer's results. Incidentally, we should like to endorse the comment made that the "écart probable" does not reflect directly the disposition made by washing of the lightest and heaviest fractions of the raw coal. From this point of view the error area has advantages as a means of expressing distribution.

Results obtained on British plants confirm the tendency noted for separation gravities, error areas and "écart probable" values to increase as the size of the particles separated decreases. The exception quoted, for separations on a concentrating table, is interesting and it would be instructive if the authors could discuss further the possible explanations which must depend on some conditions affecting washing on tables quite differently from other processes.

The section dealing with the influence of size composition is concerned simply with noting the observations on different sizes of products, but does not discuss the effect of size composition as such on the overall separation resulting from washing operations. How, for example, would the separation produced in 3 in. — 0 feed of normal composition differ from that obtained if the feed were deficient in the fine sizes? If the authors have any experience to enable them to comment on this point, it would be helpful.

There is only an incidental reference to the ash error method of expressing the results of a cleaning operation. In Great Britain, this method is often used because it expresses the effect of the misplaced material on the ash content of the clean products which is, after all, a property of great importance to the customer who buys the coal. Possibly the ash error does not add

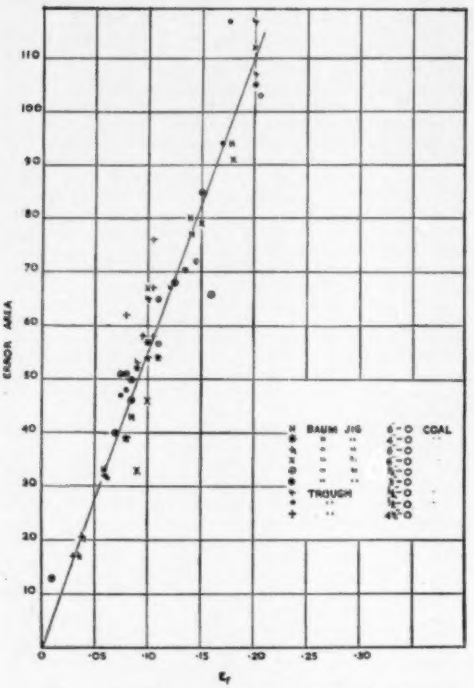

Fig. 11—Relation between error and "ecart probable."

very much to the value of distribution curves and other data illustrating the accuracy of separation from the point of view of a coal preparation engineer well versed in the various processes and methods of expressing results, but from the practical point of view it seems obvious enough that a process which can give a clean coal with an ash content differing very little from that theoretically obtainable for the same yield is more accurate than one which shows a much larger difference between the actual and theoretical ash contents. Moreover, the relation between ash error and yield error[10] enables the significance of the more accurate processes to be understood easily, especially when low gravities of separation are required. It would be interesting if the authors could comment on the use of the ash error as a means of summarising the total effect of the errors of separation.

Now that dense medium plants are being installed in relatively large numbers in the United States, it is to be hoped that much more information will be collected and published as to their performance characteristics.

K. F. Tromp *(Kerkrade, Holland)*—A number of papers on the subject of evaluating washery performance have been presented recently, but I believe the present report is the best that I have read. The authors have treated the subject in a sound and understandable way, constructively backing up their ideas with experimental results. Some of their ideas, however, are not in agreement with my own.

My concept of the distribution curve, described originally in Glückauf in 1937, is that it is composed of two parts divided by the specific gravity of separation. One part represents coal misplaced in the refuse product, while the other represents impurity misplaced in the clean coal product. The two parts are limbs of two error frequency curves which have not necessarily the same error areas. This requires some explanation. It is common knowledge that the sharpness of separation decreases when the velocity of flow through the washing unit is increased. The shorter the time available for the actual separation, the poorer the sharpness of separation, the greater the error area. This not only applies to the distribution curve as a whole but also to the constituents individually: when there is a difference in the length of time that the coal and refuse products remain in the washing unit, and mostly there is, there will be a difference in error area between the two limbs of the distribution curve.

The authors' tests on the Driessen cyclone provide an excellent example of this relation between the error areas of the two limbs of the distribution curve and the time available for the separation. Doubtless three different orifices for the spigot of the cyclone were employed in the three tests, and the total amount of medium fed to the cyclone probably remained about constant. Under these conditions, the sink or refuse material will have remained in the cyclone a bit longer in the tests made at 1.75 sp gr than in the one made at 1.56. Conversely, the float coal in the tests made at 1.75 will have remained a shorter period of time than in the test made at 1.56 sp gr. According to my theory, the error area of the misplaced float material should decrease when the float material remains longer in the washing unit, and the same applies to the misplaced sink or impurity. This is exactly the relationship shown in the following tabulation taken from Fig. 10:

Specific Gravity of Separation	Error Area Misplaced Floats	Misplaced Sinks	Total
1.56	18.7	19.3	38.0
1.63	18.1	22.9	41.0
1.75	13.7	29.4	43.1

The authors contest my thesis that the sharpness of separation decreases with increase in the density of separation. However, their cyclone tests show that the quotient of total error area and separating density is practically constant, being: 38.0 : 1.56 = 24.4, 41.0 : 1.63 = 25.1, and 43.0 : 1.75 = 24.6. This is exactly the relationship given in my aforementioned paper (Glückauf 1937) for cases where differences in viscosity can be neglected.

The other results presented by the authors in Table III, for the separations at different densities on the wet table, are not comparable because they have been carried out with different separating times. The separations at lower densities were obtained by shortening the path of travel, in other words, by shortening the time available for the particles to find their proper place. Reducing the separating time means increasing the error area. With these tests made on the wet table, two opposing factors exist, decreasing the effective table area gives an increase in error area, but the lower separating density would give a decrease in error area. Moreover, a wet table consists of a multitude of small washing units, and the distribution curve loses its definite form when the washing process consists of a series of rewashing operations.

When introducing the error curves in 1937, I stressed the point that distribution data should be calculated for only a narrow range of sizes. Otherwise, a separation according to particle size is superimposed on one according to density. The importance of this has been underestimated later by other investigators and the authors should be credited for their extensive work and their illustrative examples in demonstrating the great influence of particle size on error area and, in most processes, on separating density.

W. M. Bertholf *(Colorado Fuel & Iron Corp., Pueblo, Colo.)*—The authors have made a thorough study of the "newer" methods of evaluating coal washery performance, and this discussion is of considerable value in assisting one to understand the latest trend of thought on the subject.

It is, perhaps, unnecessary to point out that the use of "independent" criteria of washing performance requires that we work back through the original density consist of the washery feed before we have an undistorted view of the results, except in the highly improbable case of "rectilinear distribution." These criteria appear to be more useful as tools of the trade than as overall descriptive indices.

We are, therefore, inclined to agree with the authors that formulae similar to the Bureau of Mines "efficiency" (e.g. Anderson's complement of misplaced material) are more suitable for evaluating washery performance than the error area or probable error.

On page 509 of the paper a second efficiency formula is shown, involving the ash content at the specific gravity representing the most profitable separation of the coal. It is remarked that this density is difficult to determine, hence this particular formula has not been used extensively.

In the course of an investigation of the most profitable separation of coal used in the production of blast furnace coke, it was discovered that a set of "fuel cost" contour lines (per ton of pig iron) could be superimposed on the yield-ash grid ordinarily used for washability curves. Details are given in the published account[19] and the Fig. 12 is a purely hypothetical example of the use of this system in evaluating washery performance.

Taking into account the cost of the raw coal, washery operating costs and washed coal yield, we can obtain a cost per ton for any possible yield (independent of ash content). From known relationships the quantity of coke required per ton of pig iron for any ash content (independent of yield) can be computed. Combining the two, "fuel costs" for any combination of washed coal ash and yield can be developed. This grid of diagonal lines is superimposed on the rectangular coordinate system of ash vs. yield. As might be expected, the fuel cost surface slopes upward as washed coal ash increases and/or washed coal yield decreases. The yield-ash curve of the coal cuts through the fuel cost grid and reaches a minimum at some particular combination of the two variables. Going away from this point in either direction increases the fuel cost.

Considering the feed curve, which represents results obtainable only with a perfectly sharp separation, we find that the minimum fuel cost is reached at approximately 7.8 pct ash and 87 pct yield. This corresponds to a separating density of about 1.55.

The products curve, for this particular washery, indicates that in practice the minimum fuel cost will be obtained by making a separation at 9 pct ash with a yield of about 88.5 pct by weight. The separating density is 1.7 on the basis of comparable ash contents, 1.6 on the basis of comparable yields and might be 1.8 on the basis of the 50 pct point of the "distribution curve."

Note that with the imperfect separation of the actual operation the cost penalty for over-washing the coal becomes excessive and that one might just as well remove only a small proportion of the heavy impurity as to attempt to wash to 7.5 pct ash.

It is obvious that the difference between the minima on the two curves represents the cost of using an inefficient washery and that comparisons of different washing methods can be made easily for a particular coal on this three-dimensional grid. One then has a direct measure of the economic efficiency of the various methods.

It appears to be certain that captive operations methods such as those outlined above result in a very effective evaluation of coal washery performance. While the same general principles should apply to free operations, it will be much more difficult to assemble and apply the proper cost factors; nevertheless one must not lose sight of the fact that the real reason for washing coal is to reduce process costs. Proper distribution of the rewards for success along this line is a matter for discussion at another time and place.

H. F. Yancey and **M. R. Geer** (authors' reply)—The volume and scope of the discussion of this paper is most gratifying to the authors, for one of their principal objectives was to stimulate the interchange of ideas and information on this important subject.

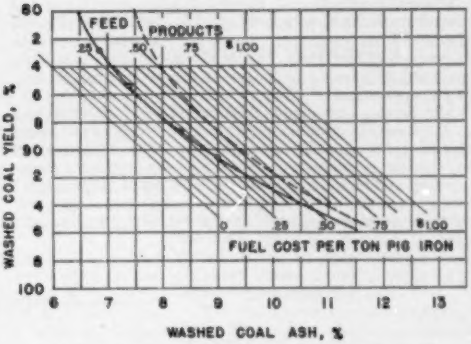

Fig. 12—Influence of washed coal ash and yield on blast furnace fuel cost per ton pig iron.

Both Messrs. Griffen and Anderson stress the importance of determining efficiency from a reconstituted or composite feed rather than a raw coal analysis to eliminate the combined effects of degradation and sampling errors. We have no argument with this thesis, except to point out that under favorable circumstances the results obtained by the two methods of computation are not significantly different. We can scarcely agree with Mr. Anderson's contention, however, that reasons of economy should ever dictate the use of his separation-efficiency formula instead of the Fraser and Yancey formula for recovery efficiency. In the first place, more densities are required to fix accurately the position of a distribution curve than are needed to provide an accurate yield-ash curve. Secondly, making float-and-sink tests of washery products without incurring the relatively small additional expense of determining the ash or sulphur content of the density fractions is stopping short of the goal. The amount of high-density impurity contaminating a washed coal is of limited interest, but the ash or sulphur content of this material is vital to both the sales department and the consumer. Similarly, the low-density fraction of the refuse is of interest to the operator only insofar as it indicates a loss of coal of saleable quality—and saleable coal is reckoned in terms of ash or sulphur content, not specific-gravity. Hence a specific-gravity analysis unaccompanied by analytical data on the products is a botchery. As Mr. Vissac points out, specific gravity is a means, not an end.

Mr. Vissac's contributions to the concept of regarding gravity separation as a probability function are of great value, but the authors cannot agree that the position of the distribution curve at specific gravities of 1.3 and 2.2 is of no practical value. With most of the bituminous coals washed in this country, the fraction lighter than 1.30 sp gr is not only the largest single density fraction but also the most valuable one. Hence the recovery effected in this density fraction is of utmost importance rather than of no practical interest. Similarly, the biggest portion of the impurity to be removed has an average specific gravity of about 2.2; how can the percentage of this material that is improperly included in the clean coal be termed of no practical interest? In fact, it may be feared that one of the hazards involved in using error area or probable error to represent a distribution curve is that this practice tends to distract attention from the curve itself. The entire curve is important, and the entire curve is needed to show the complete picture of the separation; it cannot be represented completely—and often not adequately—by one or two parameters.

Messrs. Grounds and Needham have raised several interesting questions on the role played by particle size which unfortunately we do not have sufficient data to answer completely, although Bird[18] has discussed qualitatively the importance of the finer sizes of coal in jig operations.

Mr. Tromp's treatment of some of the data contained in the paper is convincing, and doubtless no one would question the qualitative statement that sharpness of separation is inversely related to throughput. However, as stated in the paper, the published performance data examined by the authors do not show any discernible relation between sharpness of separation and density of separation. The difference in opinion on this critical point stresses the undeniable need for a great many more performance data. In fact, without more data it is difficult to judge the merits and limitations of some of the performance criteria. From this standpoint, universal adoption in the near future of the standardized performance criteria advocated by Cerchar would appear to be premature.

[17] R. E. Zimmerman: The Cleaning of Fine Sizes of Bituminous Coal by Concentrating Tables. Trans. AIME (1950) 190, p. 956; *Mining Engineering* (September 1950). TP 2875F.

[18] B. M. Bird: Chapter 22, Coal Preparation, Second Edition (1950) AIME.

High-Speed Classification and Desliming with the Liquid-Solid Cyclone

by Donald Dahlstrom

DISCUSSION

H. E. Criner *(Heyl and Patterson, Pittsburgh)*—Mr. Dahlstrom's paper is an interesting study on the use of a cyclone as a classifier. At the risk of some repetition I would like to review some material in a paper to which he refers. Although the Heyl and Patterson Co. has been primarily interested in the cyclone as an initial dewatering device a considerable amount of work has been done on the classification problem. During the study of a western Kentucky coal it was discovered that the particle sizes greater than 30 microns contained an average ash of less than 10 pct. The 20 to 30 micron material averaged about 12 pct ash, the 10 to 20 about 18 pct and around 2½ microns the ash approached 80 pct. Obviously if the +30 micron material could be separated from the remainder a desirable product, formerly wasted to the sludge pond, would result. A thickener capable of classifying at about 25 microns was chosen for this work. The thickener feed was 800 gpm at 10 pct concentration and it contained about 21 tons per hr of solids. The feed ash content averaged 15.7 pct. The material was recovered at 52 pct concentration and it contained 8 pct ash. Sixty four pct of the feed solids was recovered at the underflow. The overflow concentration was 4 pct and the average ash of the rejected material 30 pct.

Where we have been able to design the complete water circuit of a cleaning plant, the cyclones have usually been staged to produce a clean course product and a fine refuse. We have experimented with the use of extra classifying water which has the effect of making the flow ratio zero. Our classifying water was introduced to a separate smaller cyclone whose overflow was attached directly to the underflow of the larger primary cone. It was found possible to remove *all* fines below an arbitrarily chosen size by this method.

One means of steepening the classification curve, that is, to reject more material below the classification size and retain more material above that size is to successively dilute and reclassify the underflows by underflow staging. This method has the effect of squaring the cyclone recovery curve and therefore makes the overall recovery pass more steeply through the classification point. One such study was made on an anthracite slurry where it was desired to remove the −200 mesh material. The slurry solids contained 67 pct −200 mesh. The recovered material at the underflow of the second stage contained only 8 pct of the original −200 mesh and 86 pct of the +200 mesh contained in the feed.

The Cleaning of Fine Sizes of Bituminous Coals by Concentrating Tables

by R. E. Zimmerman

DISCUSSION

O. R. Lyons *(Republic Steel Corp., Cleveland)*—Some tests using wet tables conducted by an operating company to determine the effect of table-water density on the quality of the washed coal should be of interest. The only variable during the testing period was the percentage of solids in the table water.

Table operating data are shown in Table XV.

Table XV. Table Operating Data

Test No.	Average Solids in Table Water Wt, Pct	Washed Coal Ash, Pct	Washed Coal Sulphur, Pct
1	2.2	6.5	1.30
2	6.4	6.3	1.25
3	10.0	6.5	1.10

The ash content of the washed coal remained constant or nearly so, as the percentage of solids varied while the sulphur content of the washed coal decreased as the percentage of solids increased.

Copper, Zinc Allocated to Non-Communist Nations

Critical copper and zinc supplies have been allocated internationally for the first time, to stretch supplies among the 37 anti-Communist nations. The allocations will cover a three-month period, beginning October 1.

The 28-country raw materials conference awarded half of the existing supply to the United States, a total of 333,770 metric tons of copper, the remainder of the 677,160 tons divided among the other consuming nations. A total of 228,460 metric tons of zinc was allocated to the United States from the 469,260 tons for the period.

Britain was assigned the second largest amount—91,690 metric tons of copper and 60,250 tons of zinc.

Chile, one of the world's key sources of copper, hedged its acceptance with a condition that it be allowed to sell 20 pct of its output "without reference to the allocation scheme".

An announcement by the conference said all 28 member governments have accepted the proposal and have pledged to cooperate.

Atomic Age Store Opens In NYC

America's first Atomic Age department store opened in September in New York City—and already the rush is on for "Sniffers," "Snoopers," "Classmasters," "Cutie-Pies," and "Ferrets," as vari-type radioactivity detectors are called. Now the celebrated but difficult to obtain Geiger counter can be purchased across the sales counter.

Unlike the Manhattan Project the address is not secret but right on the world's busiest corner—Fifth Ave. and 42nd St. Started and staffed by several former atom bomb scientists, the Radiac Co. exists to supply everyone—prospectors, medical doctors, laboratory technicians, civil defense workers, hobbyists and engineers with a variety of atomic instruments needed in their activities. There are now Geiger counters for atomic defense, medical research, uranium prospecting, health protection against radioactive contamination from normal activities and contamination from atomic catastrophes. In the atomic defense category, there are special instruments for personnel or low-level monitoring, and for area or high-level monitoring. There are Geiger counters for atomic education.

Mining News Fronts

• Phelps-Dodge Corp. signed an agreement with the Defense Materials Procurement Agency to increase copper production from the east ore deposit of the company's Copper Queen branch at Bisbee, Ariz. The company agreed to undertake a $25 million expansion program, including construction of a concentrating and leaching plant at the Bisbee mine and enlargement of the Douglas smelter. The Government agreed to buy the first 112,500 tons of the first 150,000 tons produced at 22¢ per lb if the company could not sell at a higher price. The new facilities will have a yearly capacity of 38,000 tons and should be in production by late 1954 or early 1955.

• American Smelting & Refining Co. has started construction of a new building to house the Central Research laboratory at South Plainfield, N. J. It will have approximately 75 individual laboratories and an equal number of offices and shops. Modern apparatus to be installed will include electron microscopes, electric and fuel-fired furnaces, miniature copper refineries and zinc plants, wire drawing machinery, and large pilot plant laboratory. In the new building scientists study the base metals, and the development of new uses for a number of Asarco by-products, arsenic, and little known metals, such as indium, thallium and tellurium.

• Paradise Collieries, Inc., a newly formed subsidiary of West Virginia Coal & Coke Corp., has secured the lease rights of Pittsburgh & Midway Coal Mining Co. of western Kentucky. T. G. Gerow, president of both companies, said Paradise will begin immediate construction of a new strip mine with a cleaning plant. The mine is expected to be in operation by January 1953.

• The Defense Production Administration has approved a $46 million loan to the Harvey Machine Co. to finance construction of two new aluminum plants in the Northwest. Of this, $34 million will go for construction of a three-pot line aluminum plant near Kalispell, Mont. The remaining $12 million will be used to finance an alumina plant north of Seattle.

• Simplot Iron Mines, Inc. are now producing iron ore from their property near Elko, Nev. The ore, running 56 pct iron, is trucked 26 miles from the mine to the railhead at Palisades, Nev., where it is taken to Oakland, Calif., for shipment to Japan. The present shipment is a trial shipment of unrefined ore, and J. R. Simplot, president of the company is planning to go to Tokyo to carry on further negotiations with the Japanese company contracting for the ore.

U.S. Firms Drilling For Sulphur in Mexico

The Gulf Sulphur de Mexico, S.A. and its subsidiary Pan American Sulphur Co. are leading the exploration for sulphur which is expected to convert Mexico into one of the leading producers of the world.

This company has drilled over 50 wells to date of which over half are believed to be potential producers. Its present holdings include a 290,400 acre concession near Jaltipan, in the State of Veracruz and another of 1980 acres near Coatzacoalcos.

The companies are now negotiating for a $3½ million loan to construct a 500,000 ton-a-year sulphur extraction plant in Mexico.

Other companies investing in Mexican sulphur projects include the Mexican Gulf Sulphur Co., headed by Eugene Norton. This company is now building a 200,000 ton-a-year plant at a 1500-acre twin-dome production area known as San Cristobal Capoapan in the State of Veracruz. The more than 50 wells already drilled indicate a reserve of at least 1,500,000 tons.

Most observers in Mexico attribute this country's new sulphur boom to the faith and promotional activities of Ashley, Lawrence, and William Brady. They began a campaign to develop Mexico's sulphur in 1938 by purchasing leases from landowners, drilling test wells and interesting other companies in the properties.

International Nickel Completes Creighton Expansion

The simultaneous completion of two projects by the International Nickel Co. of Canada, Ltd. was announced by R. Leslie Beattie, vice-president and general manager of Canadian operations. The projects, a new shaft and concentrator, involved a total expenditure of $17,000,000.

The new shaft brings to 13 the number of operating shafts in International Nickel's underground mines in the Sudbury district. The new mill, which concentrates ore before transportation to the smelter at Copper Cliff, has a capacity of 10,000 tons of ore a day. The additional underground ores will serve as replacement of open-pit ores and will enable the company to continue refined nickel production capacity at the present rate of about 250,000 lb per yr.

Including these completed projects, the program has already involved expenditures of more than $100 million. When the program is completed in 1953, the company's underground mines will be able to deliver 13 million tons of ore annually, compared with 5,700,000 tons of underground ore hoisted in 1950.

The headframe and hoist house of No. 7 shaft are integral with parts of

(Continued on page 1002)

the mill building, and ore from the mine is hoisted directly into the crushing plant. The crushing plant has a simple straight-line flowsheet with no circulating loads. The grinding mills are among the largest in the world. The bulk concentrate made at the Creighton mill is pumped 7½ miles through a pipe line to Inco's reduction plant at Copper Cliff, Ont. Sand removed from the flotation tailing is returned to the mine to be used as backfill.

Coal Conversion To Fortify Fuel Supply

A great conversion-plant to fortify and coordinate U. S. fuel supplies by using the nation's immense coal reserves to produce gas of uniform quality for peak-load requirements was proposed by Frederic O. Hess, retiring president of the Gas Appliances Manufacturers Association.

Hess warned that "we do not have enough natural gas or gas lines in the East and North to solve the problem of disproportionate summer and winter supply".

"The basic concept of this operation is not merely the establishment of coal-gas plants, or standby facilities, or gasoline plants, or purely local fuel facilities," Hess said, "it is more on the order of the transformer and generating stations. It would be the TVA or the Coulee Dam of the gas industry—but would be the product of the industry, not of the government."

While not committing other industries to his conversion plant program, Hess assured the Association that the coal industry is prepared to cooperate.

The problem, he said, is basically one of coordinating research and development in all phases of the fuel industry.

It would require the cooperative efforts of experts in gas-making, coal hydrogenation, chemistry, coal mining, and the oil industry. Such a study would involve time and money, but industry-wise we are spending hundreds of thousands to survey our pipelines and millions to promote their financing.

Wales Coal Mine Gets $8 Million Coke Plant

Reports from Britain are that there may be a coal shortage this winter, but the British have not been idly waiting for winter to overtake them. One of their efforts is the opening of a new $8 million coke-oven and by-products plant to serve a coal mine at Nantgarw in Wales that was closed in 1927 because it became too difficult to work. Now by using a method called horizon mining it is being reopened and is planned to give 750,000 long tons a year. The completed plant will have a basic daily production of 1000 long tons of coke and 16,500,000 cu ft of gas to be piped to nearby Cardiff.

Nickel-Copper Strike Disclosed in Minnesota

G. M. Schwartz, director of the Minnesota Geological Survey disclosed a nickel-copper strike 8 miles south of Ely, Minn., which could relieve the national shortage of nickel if sufficient tonnage is proved.

Reportedly assaying 0.8 pct of each copper and nickel, the discovery was made on the contact of the Duluth gabbro in an area 5 miles long and of undetermined width.

A group of Duluth and Ely business men have combined to drill the property. Holes up to 100 ft have been drilled and equipment for 300 ft holes is being procured. Dr. Schwartz estimated that the low-grade strike would require tonnages of 1 million up to make ore.

Swedish Geologist Studies U.S. Iron Ore Production

Dr. Olof H. R. Odman, geologist for the Geological Survey of Sweden, is in the U.S. making a study of mining methods used in this country for the recovery of low grade deposits. Dr. Odman is investigating also modern prospecting methods, grading of ores and inventory methods.

The six-week study trip is being sponsored by the Economic Cooperation Administration under its productivity and technical assistance program.

The study is regarded as important because of the increasing demand for iron ore as the American-West European defense program is stepped up and as sources of high-grade ore are diminishing.

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aime NEWS

AIME Members Urged To Support Local Civilian Defense Activities

The announcement that Russia has tested her second atomic bomb came as no surprise to many government, military, and civilian leaders. The American public, with an 'it can't happen here' feeling, are reluctant to take the necessary steps to prepare against the possibility of an atomic attack against these shores. Although there is no direct defense against the bomb, a prepared and calm public can greatly reduce the death and chaos that would result from an attack. The Civil Defense Program can only be successful with the wholehearted cooperation of alert citizens.

AIME members are urged to engage actively in the civil defense activities in their communities. Their training should make them valuable for rescue squad training and first aid training. Local Sections can put on programs with the cooperation of Federal and State Civilian Administrations. Information kits and motion picture films are available for instruction.

R. W. Smith, AIME representative at the Civil Defense Conference in Washington, emphasized that Civil Defense is not only for cities or industrial areas. These areas will depend for help in an emergency from areas 100 or 150 miles around. Mining districts could give needed assistance in aiding neighboring cities, in times of necessity and evacuation.

Science Group Features Water Supply in Annual Meeting Symposium

The annual meeting of the American Assn. for the Advancement of Science will be held Dec. 26 to 31, 1951 at the Convention Hall in Philadelphia. Included in the six-day meeting will be a symposium of particular interest to geologists and conservationists, entitled: "The Nation's Water: Want, Waste, and Why?" This symposium will be held on Dec. 27, at 2:00 pm. Three papers will be presented at the symposium: "*Water Supply in the U. S.; Industry and Water Conservation;* and *Some Implications of Multiple Purpose River Planning.*" Discussions by authorities in each of these fields will follow presentation of the papers.

By-Law Changes Approved By Board

All changes in the by-laws published in the August issue of MINING ENGINEERING were voted by the Board of Directors at their meeting on Sept. 12 and became immediately effective. The only change made from the printed version is the addition, in Art. I, Sec. 8, of the phrase "without payment of registration fee" immediately after the words: "and be privileged to attend meetings of the Institute." In other words, no registration fee may be charged to Student Associates at any AIME meeting. Other changes voted include: Granting a credit on the initiation fee for Member or Associate Member of $2 for each year of continuous and continuing AIME membership as a Student Associate or Junior Member; standardizing the annual dues of Student Associates at $4.50, including an annual subscription to a journal; and extending the 33-year age limit for Junior Membership for those who have been in military service following World War II.

Minnesota Subsection Holds Meeting

The newly created Mining Subsection of the Minnesota Section held its second meeting on Aug. 29 at the Coates Hotel in Virginia, Minn. Mr. Earl Farnum presented a paper, *Truck Converter Units* and conducted a discussion.

The Mining Subsection was organized on May 3, 1951 to promote a better understanding of mining problems, peculiar to the Minnesota mining industry. Officers elected for 1951 were: Harry F. Kullberg, Chairman; David H. Hill, Vice-Chairman; and A. W. Kangas, Secretary-Treasurer. The next meeting will be held during November, date and place of meeting to be set.

Following the Industrial Minerals Div. fall meeting at Morgantown, W. Va., several of the more athletic repaired to the country club for golf. They were (standing from left to right): Harold Thompson, Union Carbide & Carbon; Mrs. G. F. Metz; A. B. Cummins, Chairman of the Division; Mrs. Cummins; Paul H. Price, General Chairman of the meeting; Ian Campbell, California Institute of Technology; G. F. Jenkins, Asbestos Corp.; (kneeling): H. M. Fridley, West Virginia University; E. Wick Emerson, Texas Gulf Sulphur; and G. F. Metz, Hardinge Co., who supplied the photograph.

AIME Dues Bills To Members Staggered To Facilitate Handling

Bills for 1952 AIME dues are being sent out from Institute headquarters by alphabetical groups in the closing three months of 1951. With the exception of the Petroleum Branch, those members whose names begin with "A" through "D" will be mailed their statements in October, "E" through "J" in November, "K" through "P" in early December, and "Q" through "Z" in late December. Bills to Petroleum Branch members were not sent out until after decision had been made at the Fall Meeting of the Branch regarding distribution of the statistical volume.

Although according to the bylaws the annual fee is not payable until Jan. 1, prompt payment will make it possible to issue membership cards promptly and to assure no interruption of the despatch of 1952 publications. Mailing of some 20,000 bills must, of course, be preceded by a careful check to see that the amount of each is correct, which takes some time in the aggregate. Also, the proper crediting of dues payments from the same number of members is a job of no mean magnitude, which can be accomplished much better if checks are received over a period of several months.

The statements are made out with the assumption that each member wishes to receive the same publications in 1952 as he did in 1951. If not, the member should correct the bill accordingly. Transactions volumes are billed to members at $3.50 each; annual subscriptions to one or both of the monthly journals in addition to the member's primary choice are $4 each.

Members are urged to *return the complete bill in making remittances.* When payment is made through a company or bank this is particularly necessary to identify the person to be credited, and the coverage of the payment.

Ohio Valley Section Meets

Members of the Ohio Valley Section held a meeting September 20 at the Battelle Memorial Institute, John H. Melvin presiding.

During the business session, William Smith, secretary, discussed the Morgantown, W. Va. meeting of the Industrial Minerals Div. pointing out some features of the various papers and field trips. James Cunningham, program chairman, outlined a tentative program schedule for the year, which will include two field trip meetings.

The group was given a brief talk by W. K. Sidwell of the Owens-Corning Fiberglass Corp. on the development of glass in the fiber form. Mr. Sidwell demonstrated the remarkable strength and weight properties of these materials together with the ease of fabrication using the pre-form molding techniques.

AIME Officers for 1952 Election

No supplementary nominations for AIME officers for 1952 having been received by September 1, the nominees of the official Nominating Committee are the only candidates in the field, and balloting of the Institute membership is unnecessary. Official announcement of the elections of those whose names were published in the July issue of MINING ENGINEERING will be made at the Nov. 14 meeting of the Executive and Finance Committees.

Canadian Institute Increases Dues

A 33 1/3 pct increase in dues has been voted by members of the Canadian Institute of Mining and Metallurgy, effective with 1952. The new dues are $16 for Members and Associates, and $12 for Junior Members and Junior Associates, except that a Junior Member may pay $5 for the 3-year period immediately following graduation from an approved school. The vote was 1140 for the increase, and 61 against.

Annual Meeting Paper Deadline

The Chairman of the Papers and Program Committee for the Mining Session of the annual meeting has established December 1 as the deadline for receipt of the papers, to be given at the February meeting. Papers will be acknowledged if title and a fifty-word summary or the manuscript is forwarded to E. J. Kennedy, Secretary, Technical Publications Committee, AIME, New York.

J. F. Myers To Receive Richards Award

J. F. Myers, superintendent of concentration for the Tennessee Copper Co., Copperhill, Tenn., has been selected to receive the Robert H. Richards Award for 1952. The honor will be conferred at the Institute's Annual Banquet, February 20, in New York. The Award recognizes "achievement in any form which unmistakably furthers the art of mineral dressing in any of its branches." Jack has long been active in AIME affairs, and was Chairman of the Minerals Beneficiation Div. in 1948. He is currently a member of the Board of Directors.

Geology Papers Sought

Those wishing to present papers on the Geology Subdivision program at the Annual Meeting in February 1952 should contact immediately the program chairman, George M. Schwartz, director, Minnesota Geological Survey, University of Minnesota, Minneapolis.

Error—

In listing nominations for 1952 officers of the Mineral Economics Div., on p. 617, MINING ENGINEERING, there was an error in the listing of two names. J. H. Richardson should be J. K. Richardson, and H. S. Alliston should be S. H. Williston. Both are incoming vice-chairmen of the division for 1952.

THE DRIFT OF THINGS

by Edward H. Robie

NOT often do we refer in these columns to what is going on in Washington. Several other organizations cover affairs in the national capital satisfactorily and we confess that their reports make pretty tough reading for us. The lighter stories about Washington appeal to us more, such as the one in the September issue of *Pay Dirt*, about the small mine operator who put in a hectic week going from one office to another trying to get things done. The most unusual thing that happened, he said, was when he took out a blonde the last evening he was there. Following dinner he asked for special favors, and she slapped him in the face. "You know," he said, "that was the first definite answer I had had in all that week."

But we do feel that Jim Boyd's exit from the Washington scene is worthy of note. Dr. Boyd was appointed director of the Bureau of Mines four years ago, succeeding Dr. R. R. Sayers. John L. Lewis took violent exception to his appointment and for some time one of Colorado's Senators blocked his confirmation. As a result he served without compensation for two years. Not long ago he was made head of the Defense Minerals Administration, though still keeping his title as director of the Bureau. He is reported to have differed with Interior Secretary, Oscar L. Chapman, on several matters, including payment of subsidies to mineral producers and the synthetic fuels program. He was therefore relieved of his duties and Dr. W. C. Schroeder became acting administrator. Also, many of the DMA powers were transferred to a still newer organization, the Defense Materials Procurement Agency, headed by Jess Larson. Mr. Larson has asked Howard I. Young to become his deputy in charge of the Agency's metals and minerals section, with Simon D. Strauss and Alan Bateman as advisers. Then, on Oct. 1, President Truman announced the resignation of Dr. Boyd as director of the Bureau of Mines. He has found what should be a much happier home with the Kennecott Copper Corp., which he joined Oct. 16 in an executive capacity.

Probably some political sagacity, and a willingness to follow administration policies, are required of a bureau chief in Washington, making the climate inauspicious for an independent and objective thinker and doer. However, we can at least hope that when a new head for the Bureau of Mines is chosen, he will be a man of professional stature rather than primarily a political appointee.

The Chemical Society's 75th

As this is written, in October, New York is the scene of the American Chemical Society's Diamond Jubilee celebration. Four years ago—one year late, for the AIME was founded in 1871—the AIME had a similar celebration. Our official representative at the Chemical Society's 75th birthday, Carleton C. Long, presented for us an engrossed scroll of greeting which read: "The American Institute of Mining and Metallurgical Engineers extends its most cordial congratulations to the American Chemical Society upon having so successfully attained the seventy-fifth anniversary of its founding. The Institute is proud that its roll contains the names of many members of the Society who have attained world-wide distinction in our related fields, and who have added substantially to our national industrial progress. May the Society in the future raise still further the high scientific and technical level of the chemical industry for which it is so largely responsible."

The Society now has more than 66,000 members, of whom 16,000 are chemical engineers. It is the largest scientific society in the world. It embraces 20 professional divisions, indicating the wide scope of the chemical industry. Really, almost every manufacturing industry in the country depends to a considerable extent on chemistry, as do many of the professions as well, conspicuously that of medicine. At the New York meeting 720 technical papers were presented at 129 sessions, covering 80 general subjects of scientific interest.

Ore and Near Ore

Some 8 years ago the Bureau of Mines and Geological Survey decided to classify ore reserves into *measured ore*, *indicated ore*, and *inferred ore*. The definitions were stated in the October 1943 issue of *Mining and Metallurgy*. Now comes an inquiry from England as to whether these classifications ever were accepted by mining men in this country. We referred the question to a prominent mining geologist, who writes in part as follows:

"We use these terms in our own reports but my experience with mining companies indicates that most of those in the western United States follow the old nomenclature in recording their ore reserves. Also, most of the annual reports hold to the old terms—developed (proved) ore, etc. One company qualifies further by classing ore reserves as 'fully developed ore'. An official of one of our larger companies, the American Smelting & Refining Co., said that his company uses the old terms around its operations because the operators are more familiar with them, but when dealing with the U. S. Government they are forced to use the new terms. He believed the new terms would gradually replace the old because of the insistence of the U. S. Government, if for no other reason. The Defense Minerals Administration requires the use of the terms *measured*, *indicated*, and *inferred* ore in all petitions submitted to that office.

"My own opinion is that the changeover, so far as our mining companies are concerned, will take a little time in spite of obvious advantages in the new definitions."

More Peeves

In the June issue we spoke of some words and phrases that got under our skin a bit. Robert A. Laurence, from Knoxville, Tenn., voices "a loud and hearty amen. I wish you had included two of my pet peeves: 'finalize' (to complete or finish) and 'foreseeable future.' (At what point does the unforeseeable future begin—five minutes from now, five days, five weeks, or five months?) But wasn't it President Harding who thought up 'normalcy,' for the 1920 campaign?" Yes, it surely was; we gave the credit to Coolidge.

Personals

CHARLES M. COOLEY

ALLEN H. ENGELHARDT

JOHN W. HANLEY

Allen H. Engelhardt has assumed the post of manager of operations, Cerro de Pasco Corp. He will be in charge of the mining, smelting, and refining operations in Peru. Mr. Engelhardt is replacing A. Russell Merz who is to undertake new duties in New York. John W. Hanley, superintendent of smelters and refineries, has been appointed assistant manager of operations succeeding Mr. Engelhardt.

Charles M. Cooley, assistant chief engineer of the Climax Molybdenum Co., Climax, Colo., has been engaged as assistant editor of MINING ENGINEERING. Mr. Cooley reported for duty early in October. He received a degree in metallurgical engineering from the College of Mines and Metallurgy of the University of Texas, El Paso (now Texas Western) in 1947. At college he was on the staff of the school newspaper and magazine and was president of the school engineering society. He first worked with Colorado Fuel & Iron Co. at Pueblo before joining the Climax organization 3½ years ago.

D. L. Archibald is now with the Joy Mfg. Co., St. Louis. He had been located in Mexico City for the Joy Sullivan Machinery Co. S. A.

M. W. Barlow has resigned from his position as sales manager of British Electro Metallurgical Co. Ltd., Sheffield, England to join Foundry Services Ltd., as manager of the ferroalloy div.

George D. Bellows is senior engineer with the J. G. Engineering Corp. at Djakarta, Java, Indonesia as mining consultant to the Indonesian government.

Maurice Hugh Brady is now affiliated with the Cia. Minera Asarco, Santa Barbara Unit, Chihuahua, Mexico. He had been employed by the Bunker Hill & Sullivan Mining & Concentrating Co., Kellogg, Idaho.

Philip R. Bradley, Jr., chief of the ferroalloys branch of DMA, supply div., is returning to California and will resume his practice as a private-industry consulting engnieer.

George I. Barnett is now general manager for the Las Minas de Compana S. A., Compana, Panama.

Glenville A. Collins recently returned from examining properties in British Columbia and Colorado, where the old Moro-Ajax lead-zinc mine is being reconditioned for mining after a 45-year shutdown.

J. F. Cowley has resigned as mining consultant for Eti Bank Ankara, Turkey, to become president of Vermont Copper Co., Inc., South Strafford, Vt.

C. S. Clements has become chief inspector of mines, Forestry & Geology Bldg., Fredericton, New Brunswick.

J. R. Cooper has joined the San Luis Mining Co., Tayoltita, Durango, Mexico. He had been with Cia. Minera de Alarcon, Taxco, Guerrero, Mexico.

Daniel J. Carroll has returned from Tokyo after completing 4½ years as chief coal mining engineer with General Headquarters, SCAP. He is now located in Beckley, W. Va.

Hubert O. De Beck recently accepted the appointment as consulting mining engineer, U. S. Bureau of Mines and is stationed at the southeastern regional office, Norris, Tenn. He resigned from the University of Texas.

Henry L. Day, president of Day Mines, Inc., Wallace, was elected president of the Idaho Mining Assn.

F. J. Darde is in charge of the Villaldama, N. L. unit for Compania Minera de Penoles, S.A., Mexico.

Roland I. Erickson has resigned as chief engineer and assistant superintendent with Reserve Mining Co., Babbitt, Minn. He has accepted the position of professor of mining at the University of North Dakota, Grand Forks, N. Dak.

Julian W. Feiss, formerly assistant to the director of the U. S. Bureau of Mines, has accepted the new post of executive assistant to the deputy administrator in charge of mineral procurement, DMPA, Washington, D. C.

James F. Gibbs has resigned from the Panhandle Producing & Refining Co. as manager of the land and geological dept. He is now consulting geologist and petroleum engineer in Dallas.

R. M. Haskell, superintendent of Calumet & Hecla's mills and reclamation plants has retired.

G. T. Harley, International Minerals & Chemical Corp., Carlsbad, N. Mex., has been appointed a member of the Board of Educational Finance of New Mexico.

Harry Haller is now assistant to the president, Columbia Chemical Corp., Barberton, Ohio. He had been affiliated with Pickands, Mather & Co., at the Sunday Lake Mine.

Otto Herres has returned as vice-president of Combined Metals Reduction Co., Salt Lake City. He had been working with DMA.

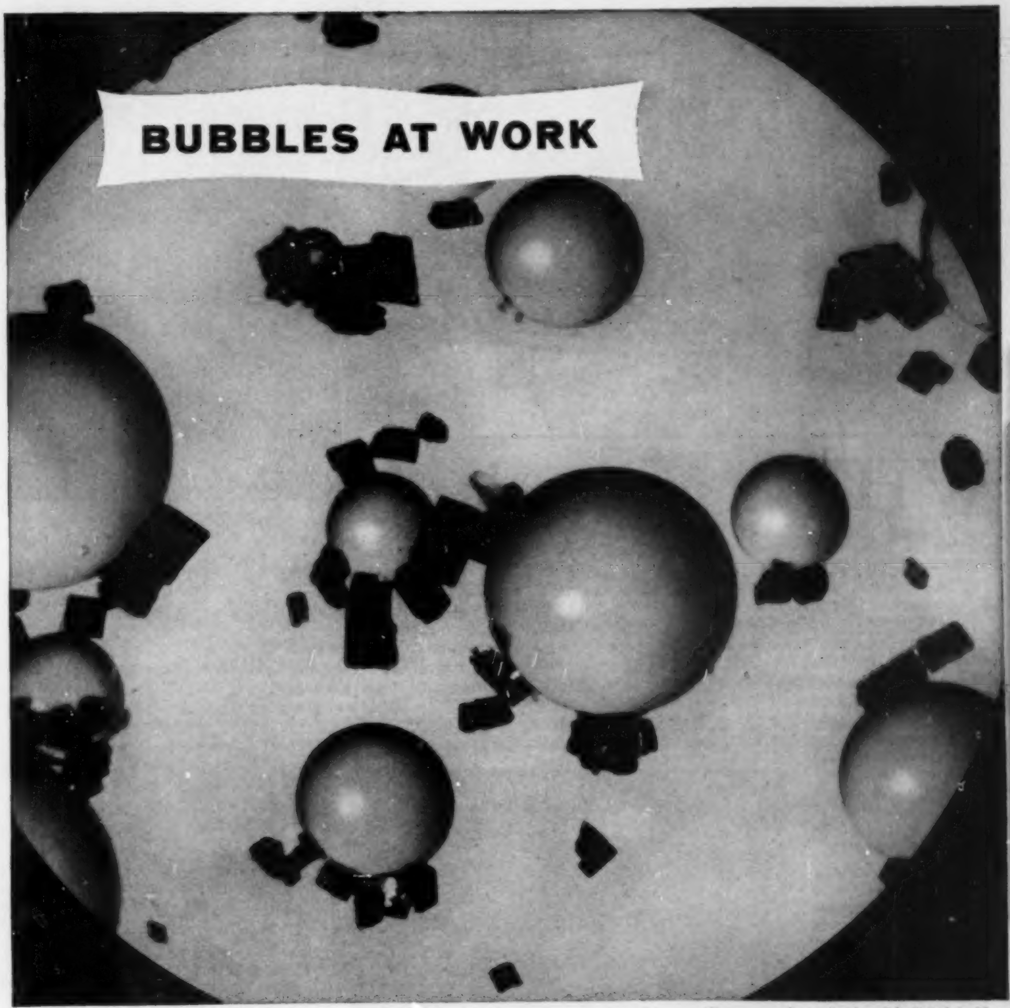
George A. Kiersch has joined the faculty of the College of Mines, University of Arizona as assistant professor of geology.

Riki Kobayashi has become assistant professor of chemical engineering at the Rice Institute, Houston.

H. A. Kursell, resident mining engineer and consultant engineer, mining dept., American Smelting & Refining Co., has retired after 26 years with the firm. He will go into private practice.

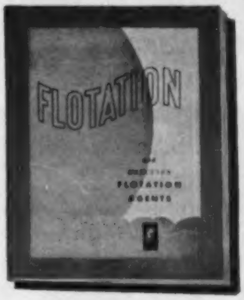
(Continued on page 1008)

BUBBLES AT WORK



Photomicrograph, greatly enlarged, by H. Rush Spedden

To help step-up flotation recoveries at low cost, Hercules has developed a number of frothers and collectors. Yarmor® F Pine Oil has been a standard of quality among pine oil frothers since the early 1920's. RADA (Hercules' Rosin Amine D Acetate) is a siliceous mineral collector that has good solubility and is easy to handle. It is used in the commercial flotation of several nonmetallic ores. These and other Hercules products are described in a 16-page technical booklet, "Flotation and Hercules Flotation Agents." This booklet also contains many formulas useful in the operation and control of modern ore-dressing mills. *Send for your copy.*



HERCULES POWDER COMPANY 955 King St., Wilmington, Delaware

HERCULES Flotation Agents

NM51-1

Robert E. Kendall is with the Resurrection Mining Co., Leadville, Colo.

Harold Kettleson has joined the Consolidated Mining & Smelting Co. of Canada, Yellowknife, Northwest Territory, Canada.

DONALD C. KIMBALL

Donald C. Kimball is now with the engineering dept., Oglebay Norton & Co., Duluth, Minn. He was formerly associated with Combined Metals Reduction Co., Stockton, Utah.

A. H. Lindley, Jr. is commodity specialist with the Bureau of Mines, Foreign Minerals Region, Washington, D. C.

Robert Levinson is in full charge of the bauxite production for N. V. Billiton Maatschappij in Surinam.

John W. Lampert has joined the engineering staff at the Resurrection Mining Co., Leadville, Colo.

James R. Miller has been appointed preparation engineer with Pond Creek Pocahontas Co., Bartley, W. Va.

Charles E. Melbye has resigned from Telluride Mines, Inc., Telluride, Colo. He has accepted a teaching fellowship at the Colorado School of Mines to work for a master's degree in geology.

David H. Orr, Jr. is now chief engineer for Phelps Dodge Corp., Morenci, Ariz.

Drury A. Pifer has been appointed a member of the State of Washington Board for registration of professional engineers.

Jack C. Pierce has been appointed assistant editor of *Compressed Air* magazine, Phillipsburg, N. J. He left the New Mexico Miners & Prospectors Assn. to take this position.

Ronald B. Pearson has been promoted to superintendent of the Canisteo mine, Coleraine, Minn. of Cleveland-Cliffs Iron Co. He had been assistant chief engineer of the Mesaba district. **Guilio Guiliani** has been named su-

(Continued on page 1010)

perintendent of the Wanless-Woodbridge mine, Buhl, Minn. He was formerly acting superintendent.

FRED A. ORLEANS

Fred A. Orleans has been appointed general manager of Compania Orleans, S. A., Mexico, and will take full charge of the firm and its interests.

Dudley L. Rainey is now with the Foote Mineral Co., Kings Mt., N. C.

George W. Robinson has been appointed general manager of Estella Mines, Ltd., Wasa, B. C.

Herbert L. Rader has accepted the position of geologist with Universal Atlas Cement Co., Allentown, Pa.

Paul M. Sorensen is now with the U. S. Tin Corp., Nome, Alaska.

William G. Sandell is now associated with the Old Hundred Gold Mining Co., Silverton, Colo., as general superintendent. He had been with the Baroid Sales Div., National Lead Co., Malvern, Ark.

Larry Toman, Jr., geologist, is with the chemical-physics branch, Squier Signal Laboratory, Fort Monmouth, N. J.

Charles E. Tonry has resigned from the Southwestern Engineering Co. to accept a position with the U. S. Bureau of Mines, Rifle, Colo.

A. J. Theis has been elected a director of Kootenay Belle Gold Mines, Ltd., Sandon, B. C. Mr. Theis had been exploration engineer and consultant for Anaconda Copper Mining Co.

James Wilbur Van Evera, Jr., has been at the Blair Limestone div., Jones & Laughlin Steel Corp., as superintendent in charge of production of both the Blair and Millville quarries since April 1950. He has the specific assignment of changing over the Blair quarry to truck haulage.

Cedric Willson has joined the staff of Texas Lightweight Aggregate Co., Dallas. Formerly he was associated with the Herculite Corp., Houston.

Robert Waskey is now with the industrial minerals branch, DMA, Washington.

Howard L. Waldron formerly field editor for *Mining World*, has joined the staff of *Engineering & Mining Journal* as associate editor.

Keith Whiting has been promoted to chief exploration engineer for the American Smelting & Refining Co., northwest div., Salt Lake City.

Obituaries

L. Douglass Anderson (Member 1914) was born on Mar. 27, 1879, in Ishpeming, Mich., and graduated in mechanical engineering from the University of Michigan in 1901. In 1905 he went to Prescott, Ariz., as construction engineer on the new smelter for the Arizona Smelting Co. He was later employed by U. S. Smelting & Refining Co. at Salt Lake City and became chief engineer for all operations in the United States, Mexico, and Alaska. After 24 years of service with that company he took a leave of absence early in 1930 to go to Australia as consulting engineer for the Mining Trust of England. Beginning his work for PCA on Aug. 1, 1933 in Carlsbad, N. Mex., Mr. Anderson was principally responsible, and a patent holder, for the flotation process of separating potash from salt. A resident of Carlsbad for 18 years, he died at the age of 72.

Wilber Judson

An Appreciation by E. C. Meagher

After a short illness, Wilber Judson died in New York on Aug. 9, 1951,

WILBER JUDSON

bringing to a close his 50 years' contribution to the minerals industry.

He was born in Lansing, Mich., on July 26, 1880, a son of James Bradford and Julia Byrnes Judson. He attended Northwestern University and the University of Michigan, but it was from Harvard University that he received his Bachelor's degree in 1901. After a year's work as an engineer in Montana, he completed his professional education at Michigan College of Mines 1903. On Aug. 28, 1917, he married May E. Reynolds, who so successfully and energetically has participated in his subsequent career.

Entering the mining business in the boom time period of Nevada, Idaho, Arizona, and northern Mexico, it is not surprising that the active and adventurous young Judson crowded into his early engineering record underground and metallurgical work in Idaho, a period as mine superintendent in Colorado, and several years in Mexico where his immediate facility with the language gave him a special interest and attachment.

After 14 years of such field work and as a result of his Mexican experience, he became associated with the W. B. Thompson interests. Newmont Mining Corp. was at that time a private company; but its early files still bear witness to the experience, quick analytical mind, and ready decision that characterized Judson's early executive career in Magma Copper Co., Mason Valley Mines Co., and New York Orient Mines Co. The technical head of these enterprises prior to World War I was Walter H. Aldridge, and mute evidence of Judson's success with these Thompson interests was recorded when he joined Mr. Aldridge with the Texas Gulf Sulphur Co. in 1921.

In 1926, he became vice-president and a director of Texas Gulf, a position ably filled until his death. His technical accomplishments were many and his broad experience, bril-

liant mind, and remarkable memory contributed to the effective discharge of his duties and to the success of the corporation that he served.

While his activity and leadership as vice-president in the American Institute of Mining & Metallurgical Engineers and the Mining & Metallurgical Society of America permit their members to claim Wilber Judson as always a miner at heart, he and May Judson for two decades also have counted many of the ablest men in the oil industry as technical comrades and warm personal friends.

Often a consultant to Government agencies, to which he gave unstintingly of his time and effort, he was returning from a meeting of the Advisory Committee on Raw Materials of the Atomic Energy Commission when he was stricken with the illness that caused his death.

We, at Texas Gulf, will miss Wilber Judson and his calm appraisal and imaginative approach to intricate problems. But to the mining industry as a whole the loss will be most felt in the future absence, wherever are gathered the people of his profession, of his ready wit, affability, charm, and the ever present halo of a truly friendly spirit.

Alfred Copeland Callen

An Appreciation by
Charles E. Lawall

Professor A. C. Callen, a member of the Institute since 1918 and head of the dept. of mining engineering at Lehigh University, died suddenly at his summer home at Ocean City, N. J. on July 30, 1951. His passing terminates a long and successful career in mineral industry education.

Professor Callen was born in Pen Argyl, Pa. on July 17, 1888. His career in mining engineering educational work began at Lehigh University where he received his undergraduate degree in mining engineering in 1909 and his master of science degree in 1911. During this period at Lehigh he also served as instructor in physics and mining engineering. After three years in industry Professor Callen went to the University of Illinois as a professor of mining engineering. In 1917 he was appointed professor of mining engineering and director of mining extension at West Virginia University, and during his tenure at West Virginia University he spent six months on a leave of absence as special agent for the Mining Federal Board for Vocational Education.

In 1924, he was appointed head of the dept. of mining engineering at the University of Illinois and remained at Illinois until he returned to his Alma Mater in 1939 as dean of the College of Engineering and head of the mining engineering dept.

During World War II he served as educational adviser for the Army Specialized Training Program in the Third Service Command.

Professor Callen was a man of many interests. While engaged in his educational work he was the author and co-author of many publications on engineering and engineering education. From 1922 to 1929 he was editor of *Coal Mine Management*. He was active in the AIME, being chairman of the Mine Ventilation Committee from 1928 to 1931; member of the Executive Committee, Coal Div., from 1932 to 1935; member of the Executive Committee of Mineral Industries Education Div. from 1934 to 1937 and again from 1940 to 1943. He was president of the West Virginia Coal Mining Institute in 1923 and president of the Illinois Mining Institute in 1930. He was much interested in the work of the Kiwanis and was governor of the Illinois-Eastern Iowa District in 1930; trustee of Kiwanis International from 1932 to 1936, and president in 1936 and 1937.

Professor Callen was active in sponsoring the annual Anthracite Conferences held at Lehigh University. He was not only the general chairman of these conferences for many years but took active part in the program and presented papers at many of them.

Professor Callen was a big man, physically, mentally, and spiritually. His successful career as an educator and as a citizen was the result of his candid understanding of the men with whom he worked, taught, and counseled. He had the uncanny ability

to analyze the capabilities of an individual better than the individual himself. His even temperament showed signs of being disturbed only when he knew that a student failed to utilize his ability to the fullest extent.

He had a true devotion to his students and time was of no essence when he had the job of serving them. He was their professor, their counselor, and their friend not only in their college years but all through their lives.

He was renowned as a citizen, an educator, and an engineer. The accomplishments and success of his former students contributed to his greatest joy and satisfaction. Today many of his former students are leading mining executives and engineers. As young men frequently select an individual whom they wish to emulate, certain it is that many of Professor Callen's students by this process of emulation acquired some of his attributes which have been factors in their success.

Professor Callen will be missed greatly by the mining industry and Lehigh University, and especially by his students but the principles he taught, the characters he molded, and the destinies he helped shape will never let them forget him.

J. P. Bickell (Member 1923) died of a heart attack on Aug. 22, 1951. Born in Molesworth, Ont., Mr. Bickell attended St. Andrew's College in Aurora, Ont. When he was 23 years old, he organized a brokerage business in Toronto. Twelve years later he retired to devote himself to his growing mining interests. At one time he was president of three mining enterprises, the largest of which was the McIntyre Porcupine Mines, Ltd. He was chairman of the board at the time of his death. During World War II Mr. Bickell made a substantial contribution to Britain's aircraft production efforts. After two years in England, he returned to Canada to head Victory Aircraft Ltd., Malton, Ont. He was a director of many firms, including International Nickel Co. and the Canadian Bank of Commerce.

Morton T. Pawel (Member 1944) died on July 3, 1951. Mr. Pawel was born at Clearfield, Pa. on Sept. 13, 1920. In 1941 he received the degree of A.B. in chemistry from Cornell University and then joined the minerals research laboratory of the Tennessee Valley Authority, Morris, Tenn. He remained here for approximately one year and then did research work on nickel for the minerals research laboratory at Muscle Shoals, Ala. for TVA. In 1942 he was engaged in work at the Bureau of Mines' laboratories and also did work at Oak Ridge and the Bureau of Standards on uranium extractions. Mr. Pawel was in the Army during World War II and following discharge he was employed by the Dorr Co., Westport, Conn. as development engineer.

William H. Munds (Member 1944) died recently. Born at Flagstaff, Ariz. in 1890, Mr. Munds graduated from the University of Arizona with the degree of B.S. After graduation he was working at a gold mine in Arizona and then was engaged in cattle business for six years. In 1924 he joined the Signal Mines Co., Yucca, Ariz. as a miner and worked his way up to mill superintendent. Mr. Munds worked for various mines in the west and in 1936 became associated with the U. S. Vanadium Corp. as assistant superintendent. He was promoted to superintendent at Bishop, Calif. In 1939 he joined the El Diablo Mining Co. Later he was made mill superintendent for Molybdenum Corp. of America at Yucca, Ariz., and then became research manager. In 1943 he joined the U. S. Bureau of Mines.

NECROLOGY

Date Elected	Name	Date of Death
1909	Robert M. Black	Sept. 10, 1951
1923	Walter Lyman Brown	Sept. 5, 1951
1918	Alfred Copeland Callen	July 30, 1951
1948	A. F. Connell	Jan. 1, 1951
1945	John F. Gallie	January 1951
1900	Leonidas C. Glenn	Jan. 11, 1950
1940	C. G. Grim	Sept. 8, 1951
1917	Garnett A. Joslin	Sept. 20, 1951
1916	Albert E. Marshall	Sept. 15, 1951
1907	F. O. Martin	June 30, 1951
1917	Hugh Park	Aug. 26, 1951
1926	Earle S. Porter	Unknown
1920	Eijiro Sagawa	1940

Metals Abstracts . . . . .

During recent months, several articles have been published in JOURNAL OF METALS, the AIME Metals Branch publication, that bear on the activities of mining site operations. These articles are listed below for the convenience of the reader.

Copies of issues in which individual articles appeared can be obtained from AIME at the single copy price of the magazine, or photostats of the articles can be obtained at the regular prices for such work.

March: **Redesigning a Secondary Smelting Plant**. Earl R. Marble, Jr. Facilities and equipment at the Newark, N. J. plant of Federated Metals Div., American Smelting & Refining Co. were redesigned and expanded to increase efficiency.

March: **El Paso Refinery of Phelps Dodge Refining Corp.** B. B. Kunkle. The history of this refinery has been one of expansion. It now has the capacity to produce 240,000 tons of cathodes annually. Flowsheets and descriptions of the plant layout, operations involved, and equipment used at one of the world's largest and most modern electrolytic copper refineries are presented in the article.

March: **Sulphur Activities in Liquid Copper Sulphides.** R. Schumann, Jr. and O. W. Moles. An equilibrium study was made of the reaction: CuS (liquid) H_2 (gas) → Cu (dissolved) H_2S (gas) at temperatures of 1150°, 1250°, and 1350°C for liquid copper sulphides ranging in composition from saturation with Cu to about 21.5 pct S. From the experimental data, activities of Cu, S, and Cu_2S in the melts were calculated. Also, the results furnish a new determination of the location of the curve showing the compositions of Cu saturated sulphide melts in the Cu-S constitution diagram.

April: **Vacuum Treatment of Parkes' Process Crusts On a Pilot-Plant Scale.** A. W. Schlechten and R. F. Doelling. Parkes' process crusts were vacuum distilled using a shortened Pidgeon retort. Zinc was effectively removed below 800°C and recovered as a zinc sheet easily stripped from the furnace liner. Lead distillation required about 950°C and the resulting lead condensate tended to stick to the thin metal liner. Final purity of residue is limited only by nonvolatile metals such as copper.

May: **Extraction of Alumina from Haiti and Jamaica Bauxites.** T. D.

Tiemann. The chemical and mineralogical composition of Caribbean bauxite ores are described. Extraction of alumina by several processes from both Haiti and Jamaica bauxites is discussed and data presented.

June: **Metallurgy of Cobalt Production from Cupriferous Pyrite.** Sanai Nakabe. Japanese wartime economy demanded domestic cobalt production. This paper describes a process operated for two years at the Besshi mine and smelter on extremely low grade (0.1 pct Co) pyrite concentrates obtained from copper ore. The steps in the process were roasting, leaching, precipitation, reduction fusion to crude cobalt, and finally refining by electrolysis to 99+ pct.

June: **Relationships Between Germanium and Cadmium in the Electrolysis of Zinc Sulphate Solutions.** S. T. Ross and J. L. Bray. The paper provides electrometallurgical data on the problem of germanium removal from zinc sulphate solutions. Germanium traces have caused much concern to the zinc refiner. Confirmatory evidence of interaction between germanium and cadmium is presented. Statistical analysis of data expands its significance and enhances its value. Further research is outlined.

July: **Separation of Copper from Zinc by Ion Exchange.** Ernest J. Breton, Jr. and A. W. Schlechten. Experiments on the separation of copper and zinc ions by selective action of ion exchange resins showed the carboxylic type to be more effective than the sulphonic resins. The latter demonstrated a greater capacity over a wider pH range. Data show the effectiveness of resins as a means of concentration.

August: **Open Hearth Charge Ore—Key to Steel Capacity Expansion.** L. B. Lindemuth.

August: **Secondary Aluminum Recovery in the South Pacific.** H. W. Franz. Aluminum from scrapped aircraft around the island of Biak was recovered by means of rather crude but effective methods.

August: **Production of Aluminum from Kalunite Alumina.** Arthur Fleischer and Julian Glasser.

October: **Metallurgical Refractories.** W. F. Rochow and J. S. McDowell. Increased metal output, improved metal quality, and increased furnace life through new developments in metallurgical refractories.

Proposed for Membership

MINING BRANCH, AIME

Total AIME membership on Sept. 30, 1951, was 17,234; in addition 2531 Student Associates were enrolled.

ADMISSIONS COMMITTEE

Thomas G. Moore, Chairman; Carroll A. Garner, Vice-Chairman; George B. Corless, F. W. Hanson, Albert J. Phillips, Lloyd C. Gibson, R. D. Mollison, John T. Sherman. Alternates: A. C. Brinker, H. W. Hitzrot, Plato Malozemoff, Ivan Given, T. D. Jones, and W. A. Clark, Jr.

Institute members are urged to review this list as soon as the issue is received and immediately write the Secretary's Office, night message collect, if objection is offered to the admission of any applicant. Details of the objection should follow by air mail. The Institute desires to extend its privileges to every person to whom it can be of service but does not desire to admit persons unless they are qualified. Objections must be received before the 30th of the month on Metals and Mining Branches.

In the following list C/S means change of status; R, reinstatement; M, Member; J, Junior Member; A, Associate Member; S, Student Associate.

Alabama
Fairfield—Burch, Elza F. (J) (R. C/S—S-J)

Arizona
Morenci—Culhane, Frank R. (J)
Morenci—Ludden, Robert B. (J)
Phoenix—McNabb, Donald G. (A) (R. C/S—S-A)
Ray—Ballmer, Ray W. (J) (R. C/S—S-J)
Tucson—Vought, Leland C. (J) (C/S—S-J)

California
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San Gabriel—Miller, Robert P. (M)

Colorado
Climax—Conwell, James A. (M)

Georgia
Milledgeville—Falvey, Arthur E. (M) (R. C/S—S-M)

Idaho
Soda Springs—Emigh, George D. (M)
Soda Springs—Gurvin, John E. (M)

Illinois
Hinsdale—Rawlins, Roderick K., Jr. (J) (R. C/S—S-J)

Maryland
Seat Pleasant—Weiss, Edwin M. (J)

Minnesota
Minneapolis—Chang, Chen-Siang (J) (C/S—S-J)
Virginia—Iverson, Cedric L. (J) (R. C/S—S-J)

Missouri
Rolla—Gabriel, V. G. (M)
Rolla—Miller, Robert J. (J)

Montana
Butte—Ingersoll, Robert G. (J) (C/S—S-J)
Butte—McQuay, Ray (M)

Nevada
Ruth—Goring, Arthur W. (M) (R. C/S—J-M)
Winnemucca—Scott, James B. (J) (C/S—S-J)

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Carlsbad—Rasor, James H. (M)
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Santa Rita—Ward, John, Jr. (M)
Silver City—White, Athington (M) (R. C/S—J-M)
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Vanadium—Kirchman, Robert C. (M)
Vanadium—Moser, George L. (M)
Vanadium—Parson, J. M. (M)

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Charleston—School, Richard, Jr. (A)

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Ducktown—Kingman, Owen (M) (R. C/S—S-M)

Utah
Lark—Gillam, Clyde W. (M)
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Palmer—Galli, Peter E., Jr. (J) (C/S—S-J)

Australia
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Canada
Montreal—Desrosiers, Paul H. (A)

Central Africa
Ruanda—Van der Klugt, Simon (M)

England
Surrey—Browne, Edward H. (M)

Japan
Fukuoka—Ken-Kizaki, Fumio (M)

Mexico
Durango—Croze, Joseph B. (M)
Mexico D.F.—Rood, Norman P. (A)
Monterrey—Lee, Robert L. (A)

—Coming Events—

Nov. 2, Illinois Mining Institute, annual meeting, Hotel Abraham Lincoln, Springfield, Ill.

Nov. 6, AIME, Boston Section, MIT, Campus Room, Graduate House, Cambridge, Mass.

Nov. 6, AIME, Morenci Subsection, Longfellow Inn, Morenci, Ariz.

Nov. 7, AIME, Chicago Section, Chicago Bar Assn., Chicago.

Nov. 8, AIME, St. Louis Section, Engineers Club, St. Louis, Mo.

Nov. 8-9, Joint Engineering Societies' Symposium on Corrosion, Rodger Young Auditorium and Biltmore Hotel, Los Angeles.

Nov. 9, American Iron & Steel Institute, regional technical meeting, Hotel Mark Hopkins, San Francisco.

Nov. 14, AIME, San Francisco Section, students' meeting, Stanford University, Stanford, Calif.

Nov. 15, AIME, Utah Section, Hotel Newhouse, Salt Lake City.

Nov. 15-16, Magnesium Assn., annual meeting, Biltmore Hotel, New York.

Nov. 15-17, International Mining Days, El Paso, Texas.

Nov. 25-30, ASME, annual meeting, Chalfonte-Haddon Hall, Atlantic City, N. J.

Nov. 26-30, ASME, annual meeting, Chalfonte-Haddon Hall, Atlantic City, N. J.

Nov. 26-Dec. 1, Chemical Industries Exposition, Grand Central Palace, New York.

Nov. 28-30, Scientific Apparatus Makers Assn., midyear meeting, laboratory apparatus, optical, nautical, aeronautical, and military instrument sections, Hotel New Yorker, New York.

Nov. 29, American Iron & Steel Institute, regional technical meeting, Hotel Cleveland, Cleveland.

Dec. 2-5, American Institute of Chemical Engineers, annual meeting, Chalfonte-Haddon Hall, Atlantic City, N. J.

Dec. 5, AIME, Chicago Section, Chicago Bar Assn., 29 S. La Salle St., Chicago.

Dec. 6-8, AIME, Electric Furnace Steel Conference, William Penn Hotel, Pittsburgh.

Dec. 7, AIME, Lehigh Valley Section, annual meeting, Hotel Bethlehem, Bethlehem, Pa.

Dec. 10, AIME, Arizona Section, annual meeting, Pioneer Hotel, Tucson.

Dec. 11, AIME, Morenci Sub-section, Longfellow Inn, Morenci, Ariz.

Dec. 27-28, American Chemical Society, Div. of Industrial & Engineering Chemistry, symposium on nucleation, Northwestern University, Evanston, (Chicago).

Jan. 9, 1952, AIME, Chicago Section, Chicago Bar Assn., 29 S. La Salle St., Chicago.

Jan. 16-18, Society of Plastic Engineers, Inc., annual national technical conference, Edgewater Beach Hotel, Chicago.

Feb. 6, AIME, Chicago Section, Chicago Bar Assn., 29 S. La Salle St., Chicago.

Feb. 18-21, AIME, annual meeting, Hotel Statler, New York.

Mar. 3-7, ASTM, spring meeting and committee week, Hotel Statler, Cleveland.

Mar. 5, AIME, Chicago Section, Ladies' Night, Chicago.

Apr. 25-26, AIME, New England Regional Conference, Kenmore Hotel, Boston.

May 6-9, Scientific Apparatus Makers Assn., annual meeting, Edgewater Beach Hotel, Chicago.

May 22-24, American Society for Quality Control, annual convention, Onondaga County War Memorial, Syracuse, N. Y.

June 23-27, ASTM, 50th anniversary meeting, Hotel Statler, New York.

July 1-Sept. 30, Centennial of Engineering, Chicago.

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